RI 9580

REPORT OF INVESTIGATIONS/1995

Longwall Gate Road Stability in a Steeply Pitching Thick Coal Seam With a Weak Roof

UNITED STATES DEPARTMENT OF THE INTERIOR

UNITED STATES BUREAU OF MINES

U.S. Department of the Interior Mission Statement

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally-owned public lands and natural resources. This includes fostering sound use of our land and water resources; protecting our fish, wildlife, and biological diversity; preserving the environmental and cultural values of our national parks and historical places; and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to ensure that their development is in the best interests of all our people by encouraging stewardship and citizen participation in their care. The Department also has a major responsibility for American Indian reservation communities and for people who live in island territories under U.S. administration.

Cover: USBM engineers install BPC's at gate road study site, Shoshone No. 1 Mine. (Photo: Mark Larson, Spokane Research Center, U.S. Bureau of Mines)

Report of Investigations 9580

Longwall Gate Road Stability in a Steeply Pitching Thick Coal Seam With a Weak Roof

By Lance R. Barron and Matthew J. DeMarco

UNITED STATES DEPARTMENT OF THE INTERIOR
Bruce Babbitt, Secretary

BUREAU OF MINES
Rhea Lydia Graham, Director

International Standard Serial Number
ISSN 1066-5552

CONTENTS

ILLUSTRATIONS

ILLUSTRATIONS—Continued

TABLE

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

Metric Units

cm	centimeter	m	meter
kg/m^3	kilogram per cubic meter	mm	millimeter
km	kilometer	MPa	megapascal
kPa	kilopascal	t/m	metric ton per meter

U.S. Customary Units

ft	foot	psi	pound (force) per square inch
in	inch	psi/ft	pound per square inch per foot
pcf	pound (force) per cubic foot	st/ft	short ton per foot

LONGWALL GATE ROAD STABILITY IN A STEEPLY PITCHING THICK COAL SEAM WITH A WEAK ROOF

By Lance R. Barron[1] and Matthew J. DeMarco[1]

ABSTRACT

The U.S. Bureau of Mines (USBM) conducted ground pressure analysis of a wide abutment-type chain pillar in a two-entry gate road of a Western U.S. coal mine with an extremely weak immediate roof. About 15 m of fragile, low-strength mudstone lies between the seam and the lowest competent roof member. Three- and two-entry gate road designs with several pillar sizes and various secondary support systems have been employed to improve tailgate-entry stability, with varying results.

This report discusses gate road layout and performance and secondary support effectiveness. The results of the pillar pressure study are compared to pillar loading predicted by a widely used pillar design method and to similar studies in other mines. A stability evaluation of the most recent longwall headgate, using the USBM Analysis of Longwall Pillar Stability (ALPS) method, indicates marginal stability in first-panel mining and instability in second-panel mining. The ALPS method and the USBM Coal Mine Roof Rating system are used to evaluate tailgate-mining stability of the previous gate roads and to determine pillar and entry width and top coal thickness criteria for tailgate stability in future panels.

[1]Mining engineer, Denver Research Center, U.S. Bureau of Mines, Denver, CO.

INTRODUCTION

The U.S. Bureau of Mines (USBM) has conducted an ongoing effort to improve safety and efficiency in longwall mining operations. Of particular concern are the gate roads used to access longwall panels during all phases of mining. Many field studies of gate road stability have been carried out in mines throughout the United States for a wide range of mining depths, ground conditions, and panel and gate road layouts. The results of these studies, together with numerous case histories of gate road design and longwall mining experience, form an extensive database for developing gate road design methods and for evaluating gate road performance. Field studies and case histories of atypical gate road designs and/or unusual mining conditions serve to expand the database, elevate the state of the art in gate road design, and provide performance-based guidelines for designing future gate roads.

The chain pillars in longwall gate roads are of two basic types: abutment pillars, designed for withstanding all loads to which they are subjected throughout longwall mining operations; and yield pillars, designed to gradually crush under load that is then transferred onto nearby supporting pillars, panel blocks, and/or mined-out areas. Common U.S. gate road design practice is to employ only yield pillars in two-entry gate roads, whereas abutment pillars, either alone or in combination with yield pillars, are utilized in multientry (three or more) gate roads. The Shoshone No. 1 Mine has, however, used six two-entry gate roads with abutment pillars (accessing the last five longwall panels) in efforts to reduce the likelihood of spontaneous combustion and to improve gate road structural stability for an extremely weak roof and increasing mining depths.

During 1993 and 1994, the USBM conducted a gate road stability study in the Shoshone No. 1 Mine. During the study, various gate road design parameters, such as overall gate design, gate road width, pillar sizes, entry widths, and support requirements, were investigated and correlated with intrinsic ground conditions, such as cover depth; geologic structure and stratigraphy; lithology and physical properties of the coal, roof, and floor; and loading attributable to nearby mined-out areas. The performance of the two basic gate road designs used at the mine was then evaluated according to their relative success in meeting the operator's requirements for stability, safety, and longwall productivity. The results of the study, in conjunction with the history of the operator's efforts in developing the gate roads and mining the adjacent longwall panels, constitute an important background for developing gate road design criteria and procedures for future longwall operations.

LOCATION OF STUDY SITE

The Shoshone No. 1 Mine is located approximately 6.4 km (4 miles) north-northeast of the town of Hanna in Carbon County, south-central Wyoming. The mine portals and surface facilities are accessed by a 1.3-km (0.8-mile) improved road that intersects County Road 291 about 5.6 km (3.5 miles) north-northeast of Hanna (figure 1).

GEOLOGIC SETTING OF STUDY SITE

The Shoshone No. 1 Mine is situated in the northeastern part of the Hanna Basin, which, together with the Carbon Basin to the southeast, constitutes the Hanna Coalfield (*1*).[2] The intermontane Hanna Basin, formed during the Laramide Orogeny about 38 to 65 million years ago (*1*) (Early Tertiary time), is an asymmetrical structural basin over 16.1 km (10 miles) across, with a total thickness of nearly 10,000 m (33,000 ft) of sediments overlying its deepest portion in the northeasterly plunging Hanna Syncline (*2*).

[2]Italic number in parentheses refer to items in the list of references at the end of this report.

STRUCTURE

Folds

The Hanna Basin is separated into two synclines whose axes are nearly perpendicular to one another. The southernmost is called the Hanna Syncline, whereas the northernmost is unnamed (*1*). The Shoshone No. 1 Mine lies on the southwestern flank of the unnamed northern syncline (figure 2) (*3*); consequently, the strata in the mine area dip approximately 16° to the northeast (*4*).

Faults

Three faults are present in the mine area, as shown in figure 2. The Red Hills Fault, a northwest-southeast-trending reverse fault with vertical displacements ranging from 7.6 to 9.1 m (25 to 30 ft) (*5*), lies to the southwest of the mine workings. The Dixie Draw Fault, a northeast-southwest-trending normal fault with approximately 11.3 m (37 ft) of measured vertical displacement (*5*), is located to the southeast of the mine workings. The Barrel Springs Fault has the greatest effect on the overall mine layout and bounds the farthest downdip extent of the main entries of the mine. This normal fault trends approximately N. 30° W. in the mine area and dips about 70° to the northeast (*3*). The strata on the northeastern side of this fault have been downthrown an average of 21 m (70 ft) (*5*), with total vertical displacement ranging from 4.6 m (15 ft) in the southeastern part of the mine area to 30.5 m (100 ft) in the northern part (*2*).

Joints

Two primary, well-developed joint sets (measured at the outcrops and on oriented drill cores), one oriented N. 25° to 40° E. and the other oriented N. 40° to 55° W., are present in the mine area (*5*). A third, less-developed set, measured near the Barrel Springs Fault, is oriented N. 80° E. (*5*). All three sets correlate with jointing measured in coal-bearing strata throughout the Hanna Basin (*1*).

STRATIGRAPHIC UNITS

In contrast to the coal-bearing strata of other basinal coal regions of the central and southern Rocky Mountains (such as the Wasatch Plateau of Utah, the Uinta Basin of Utah-Colorado, the Green River Basin of Colorado-Wyoming, and the Raton Basin of Colorado-New Mexico), which were generally deposited during the Late (Upper) Cretaceous in a near-marine delta-floodplain-estuary environment (*6-7*), most of the economically important coal-bearing strata of the Hanna Basin were deposited during the Early (Lower) Tertiary in a continental environment of floodplains, alluvial fans, and braided streams near the shorelines of freshwater lakes in the intermontane basin formed by the Laramide uplift of the surrounding mountain ranges (*1*).

In ascending order, the principal stratigraphic units of the Hanna Basin are the Mesaverde Group (composed of the Haystack Mountain Formation, Allen Ridge Formation, Pine Ridge Sandstone, and Almond Formation), the Lewis Shale, and the Medicine Bow, Ferris, and Hanna Formations (*8*) (figure 3).

Lithology of Coal-Bearing Formations

The most significant coalbeds occur in the Ferris and Hanna Formations, which generally occupy the north-central portion of the Hanna Basin (*1*). According to Glass and Roberts (*1*):

> Rocks associated with the Ferris and Hanna formation coals are perhaps the most variable in the coal field. Massive, cross-bedded, sometimes conglomeratic sandstone units are interpreted as fluvial in origin, deposited by meandering or braided streams. These units are lenticular in cross section and linear or sinuous in plan view. Although they frequently crop out as impressive cliffs, in reality they are not the dominant lithology of the coal-bearing rocks. By far, the finer grained siltstones, claystones, and shales predominate. These fine grained rocks are variously interpreted as overbank deposits laid down during flooding of the major fluvial channels or lacustrine in origin, indicating the existence of fresh-water lakes or ponds.
>
> Commonly, very dirty coals or interbedded coal and shale units overlie some of the coals. These units, variously called coaly shale or carbonaceous shales [or mudstones], are extremely high in ash (greater than 30%) and not coals in a strict sense.
>
> All these various lithologies may abut or grade into one another over very short distances or vertical intervals. In the case of the sandstones, abrupt angular contacts with adjacent units are common where sandstones now fill erosional channels cut by the major streams that flowed into the Hanna Basin from surrounding highland areas and then eastward out of the coal field.

Hanna No. 80 Coalbed and Adjacent Strata

Although the 732- to 2,438-m (2,400- to 8,000-ft) thick Hanna Formation (*1*) contains at least 30 seams in the mine area, only 3 are of economic thickness (*9*). Figure 4 shows stratigraphic columns of the upper portion of the Hanna Formation (*2*) from two drill holes, DH94 and DH104, located in the northern part of the mine area (figure 5). The Shoshone No. 1 Mine extracts the Hanna No. 80 Coalbed (*2, 5*). The Hanna No. 82 Coalbed, which lies about 91 to 152 m (300 to 500 ft) above the No. 80, has been mined by surface methods over much of the southwestern (updip) portion of the Shoshone No. 1 Mine, whereas the Hanna No. 84 Coalbed, about 100 m (330 ft) above the No. 82 and 200 to 260 m (660 to 850 ft) above the No. 80, has not been mined above the Shoshone workings. No known mine workings underlie the No. 80 Coalbed on the mine property.

Hanna No. 80 Coalbed

The Hanna No. 80 Coalbed ranges from 4.3 to 4.9 m (14 to 16 ft) thick in the mine area (*2, 4, 9*) and exhibits pronounced calcite-filled cleating, with face cleat spacing ranging from 0.16 to 0.32 cm (1/16 to 1/8 in) and butt cleat spacing ranging from 30.5 to 61 cm (12 to 24 in) (*5*). A 2.5- to 7.6-cm (1- to 2-in) carbonaceous shale or mudstone parting about 1.2 to 1.8 m (4 to 6 ft) above the base of the bed is persistent in the northern portions of the mine, while to the south and west, the number and thickness of noncoal partings increase. Most of the partings, however, are not laterally persistent and pinch out rapidly to the north and east (*2*). In common with many of the coalbeds of the Hanna Coalfield (*1*), the Hanna No. 80 Coalbed is highly prone to spontaneous combustion.

Roof and Floor Strata

The strata immediately overlying the Hanna No. 80 Coalbed (the immediate roof) consist of a highly laminated and slickensided mudstone, persistent throughout the mine area, that varies from 2.5 to 5.2 m (8 to 17 ft) in thickness (*10*) and is highly friable (*9*) and moderately moisture sensitive (*10*). The mudstone is overlain by a 0.6- to 6-m (2- to 20-ft) thick siltstone (*10*) (figure 6). The immediate mine floor consists of a highly moisture-sensitive, gray-to-buff mudstone. The remainder of the overburden (the main roof) and the strata underlying the floor mudstone consist of the various rocks and lithologies typical of the Hanna Formation (figure 4).

Physical Properties

Physical properties testing of Hanna No. 80 coal and adjacent roof and floor rock was performed from 1976 to 1977, prior to opening the mine, as part of a geotechnical site investigation and rock mechanics study (conducted by geotechnical consultants retained by the initial owner-operator) to determine appropriate mine layout and gate road system designs for longwall mining (*5*). Samples for testing were obtained primarily from core drilled from the surface at numerous locations across the property (*5*). Because of poor recovery of intact core (owing to the highly fragmented nature of the rock), however, additional samples were obtained from the highwall of an abandoned strip pit (surface mine) at the Hanna No. 80 outcrop.[3]

In 1992, physical properties testing of coal and roof rock obtained from the longwall area of the Shoshone No. 1 Mine was conducted by the USBM. Again, owing to the characteristics of the mudstone, all intact roof rock core specimens obtained for testing were of insufficient length for measurement of axial deflection and excessively fractured and fragmented for measurement of lateral deflection; therefore, neither elastic modulus nor Poisson's ratio values could be determined for the immediate roof mudstone. Twenty coal core samples suitable for testing were drilled from the available bulk specimens and used in uniaxial and triaxial tests.[4]

A compilation of the average physical properties values, determined through both testing programs, for the Hanna No. 80 coal and the immediate roof and floor rock is presented in table 1. Using the rock mass classification system developed by Bieniawski (*11*), together with the uniaxial compressive strength values determined in the 1976-77 testing, Djahanguiri (*5*) reported the Hanna No. 80 coal as ranging from very low to low strength, the floor mudstone (or carbonaceous shale) and sandstone as ranging from low to medium strength, and the roof mudstone (or carbonaceous shale) as ranging from very low to medium strength. When the Bieniawski classification is utilized with the 1992 testing results, the uniaxial compressive strength of the coal again ranges from very low to low; however, all roof mudstone samples tested are within the very low-strength category.

[3]Private communication from F. Djahanguiri, USBM (formerly of Dravo Corp.), 1994.

[4]Results of physical properties testing of Shoshone Mine coal and roof rock. Letter report from S. C. Tadolini, USBM, to A. P. Schissler, Cyprus Coal Co., May 12, 1992, 7 pp.

Table 1.—Physical properties of Hanna No. 80 Coalbed and adjacent strata

Rock type	Average uniaxial compressive strength		Average elastic modulus		Average Poisson's ratio	Shear strength		Angle of internal friction, (ϕ), deg
	MPa	psi	MPa	10^6 psi		MPa	psi	
Coal[1]	8.8	1,270	3,310	0.48	0.52	3.0	441	41
Coal[2]	17.7	2,568	3,448	0.50	0.46	3.5	510	50
Floor mudstone[1]	39.6	5,740	8,000	1.16	0.34	19.4	2,806	NA
Floor sandstone[1]	28.1	4,070	7,724	1.12	0.42	5.9	858	44
Roof mudstone[1]	24.3	3,530	4,965	0.72	0.27	6.5	939	41
Roof mudstone[2]	9.7	1,410	NA	NA	NA	NA	NA	NA

NA Not available.
[1]Physical properties testing conducted from 1976 to 1977. Source: Djahanguiri (5, p. 1C5-4).
[2]Physical properties testing conducted in 1992 at the USBM's Denver Research Center.

MINING OPERATIONS AND CONDITIONS

COAL MINING IN HANNA BASIN

From 1888, when mining in the Hanna Basin began, until 1954, when the last of the Union Pacific Coal Co. (a subsidiary of the Union Pacific Railway Co.) mines was closed, nearly 33 million tons of coal was produced from 14 underground mines (8 Union Pacific mines and 6 others), most of which were located in the southern part of the basin (*12*). Mine disasters, such as floods, fires, and explosions, caused numerous fatalities in these early operations (*12*). Additionally, roof and rib falls, some of which were associated with bumps (*12*) probably caused by the sudden failure of overloaded remnant pillars in high-extraction pillar-robbing operations, were frequent hazards.[5] From 1954 until the opening of the Carbon No. 1 Mine (later renamed as the Shoshone No. 1 Mine), the only underground coal mines in the basin were Energy Development Co.'s Vanguard Mines about 8.8 km (5.5 miles) west of Hanna (*1*)—the Nos. 1 and 3 Mines that operated from 1971 to 1973 (*13*) in the No. 65 and Brooks Coalbeds of the Ferris Formation (*12*), and the No. 2 Mine that operated from 1974 to 1983 (*13*) in the No. 50 Coalbed (also Ferris Formation) (*1, 14*).

Strip (surface) mining in the Hanna Basin began in 1937 and has accounted for most of the coal production from the area since 1954 (*1, 12*). In 1993, the only producing coal mine in Carbon County other than the Shoshone No. 1 Mine was the Medicine Bow strip mine, operated by Arch Minerals Corp. (*13*) at the western flank of the basin about 19.2 km (12 miles) northwest of Hanna (*1*).

SHOSHONE NO. 1 MINE OPERATING HISTORY

The Shoshone No. 1 Mine, originally named the Carbon No. 1 Mine, was opened in 1979 by Carbon County Coal Co. (CCCC), a joint venture of Rocky Mountain Energy Co. and Dravo Corp., the operating partner (*2, 15*). The mine was closed by CCCC in 1986 (*16*) and acquired in 1987 by the current owner-operator, Cyprus Shoshone Coal Corp. (CSCC), a subsidiary of Cyprus Coal Co. (*17*).

From the outset, the mine was designed and developed for retreat longwalls as the principal means of production. CCCC developed the portals in the 50-m (165-ft) highwall of the abandoned Rosebud Coal Sales Co. Pit 4 strip mine (*2, 15*) (figure 6) and drove the five-entry mains approximately 2,000 m (6,600 ft) downslope (subparallel to the seam dip) until nearly encountering the Barrel Springs Fault (figure 5). From that point, the main entries were extended in a 518-m (1,700-ft) offset along the seam strike, then developed another 152 m (500 ft) downslope (parallel to the original orientation) until termination near the Barrel Springs Fault. In accordance with the pillar sizes recommended in the predevelopment geotechnical study (*5*), the chain pillars in the main entries were initially sized 25 by 25 m (82 by 82 ft) with 5.5-m (18-ft) entry and crosscut widths (*15*). In much of the offset and downslope extension, however, pillar length was increased to 50 m (164 ft) in response to increased mining depth and also to reduce the number of roof-fall-prone intersections.[6]

Longwall Gate Roads

Carbon County Coal Co. Development

CCCC completely developed the 1st to 6th Left (figure 5) and 1st Right (outside the area shown in figure 5) gate roads. Although CCCC initially planned to use a two-entry gate road design, a lengthy delay in receiving the required regulatory variance (for a two-entry system) necessitated development of the first five gate roads as three-entry systems.[7] In the 1st to 5th Left gate roads, pillar widths were incrementally increased from 17.7 m (58 ft) in 1st Left (*15*) [also developed in accordance with the geotechnical study recommendations (*5*)] to 24 m (80 ft) in 5th Left, primarily in response to increasing mining depth. Small room-and-pillar sections (with no pillar extraction) were developed from 1st to 3rd Left (figure 5) and 1st Right in small areas isolated between these gate roads and abandoned strip mines that bounded the west and northwest sides of the mine reserves. CCCC discontinued this practice because of concern that air might "leak" into these pillar workings from the nearby No. 80 seam "outcrops" in the old strip mines and possibly contribute to spontaneous combustion problems.[8] Since 1985, continuous miners have only been used for gate road development.[9] CCCC developed the 6th Left gate road as a two-entry system to minimize the amount of coal remaining in the gob area after the panel was sealed and thus reduce the potential for spontaneous combustion.[10]

Cyprus Shoshone Coal Corp. Development

CSCC developed the 7th to 11th Left gate roads from 1989 to 1993. Since final regulatory variance permitting

[5]Private communication from G. B. Glass, Wyoming State Geologist, 1994.

[6]Private communication from D. E. Routon, Mine Engineers Inc. (former chief engineer, Carbon County Coal Co.), 1994.

[7]Longwall Gate Road Performance Summary. Letter report from C. L. Stewart, Cyprus Shoshone Coal Corp., to L. R. Barron, USBM, May 3, 1995, 3 pp.

[8]Source cited in footnote 6.

[9]Coal Mine Roof Rating Study at Shoshone Mine. Letter report from C. Mark, USBM, to K. Williamson, Cyprus Shoshone Coal Corp., Sept. 3, 1993, 8 pp.

[10]Source cited in footnote 6.

the use of two-entry gate roads had been received, the 7th to 9th Left gate roads were developed as two-entry systems to reduce the extent of exposed roof and secondary support requirements during longwall panel retreat.[11] Chain pillar width was reduced from 24 m (80 ft) in 7th Left to 21 m (70 ft) in 8th Left, then increased back to 24 m in 9th Left. To inhibit spontaneous combustion and improve tailgate ground conditions, a 24-m (80-ft) barrier pillar was retained between 7th and 6th Left to isolate the extensive gob area of the 1st to 5th Left longwall panels (figure 5) from air ventilating the active workings and also to minimize loads transferred from this large mined-out area onto the 7th Left gate road.[12]

The first 640 m (2,100 ft) of the 10th Left gate road was developed by CCCC as a two-entry system with 24-m (80-ft) wide chain pillars. CSCC completed development utilizing this design, but with angled crosscuts to facilitate shuttle car haulage during entry drivage.

CCCC developed the first 488 m (1,600 ft) of the 11th Left gate road utilizing a three-entry configuration with 24-m (80-ft) wide chain pillars and "staggered" crosscut centers to eliminate four-way intersections. When CSCC resumed gate road development, this design was modified to a two-entry configuration with 30.5-m (100-ft) wide pillars and angled crosscuts. This design was used for the remaining development of the gate road.[13]

Retreat Longwall Panels

Nine longwall panels have been developed and fully extracted in the Shoshone No. 1 Mine (figure 5), and extraction of the tenth panel (outside the area shown in figure 5) is currently under way. The nine panels were consecutively extracted downdip, beginning with the portion of the mine nearest the seam outcrop. In each panel, the headgate was always located on the downdip side to facilitate cutting and loading of the coal; therefore, the tailgate was always located on the updip side of the panel adjacent to the gob of the preceding mined-out panel.[14]

Beginning in 1981 (*2, 15*), the first four longwall panels, 1st to 4th Left, were extracted by CCCC. The panel width (face length) for all four panels was 122 m (400 ft), and the lengths of the panels were determined by the reserve boundary at the limits of the abandoned strip mines. Mining depths for these panels were 122 m (400 ft) or less (figure 5), and ground conditions were generally good, with minimal secondary support requirements in the tailgates.[15]

Panel width was increased to 183 m (600 ft) in the fifth panel, 5th Left. The first (inby) half of this panel was extracted by CCCC before mine closure in 1986, and CSCC mined the remainder of the panel in 1989.[16] Ground conditions in the three-entry tailgate were generally good during the first three-quarters of panel retreat. However, in the last 457 m (1,500 ft) of the panel, tailgate conditions quickly deteriorated, with massive roof falls repeatedly occurring well in advance of the face, thus necessitating drivage of a new (fourth) tailgate entry to complete panel extraction.[17]

CSCC extracted the 8th, 9th, and 10th Left panels (figure 5) from 1989 to 1993. Ground control, particularly in the tailgates, became progressively more difficult as mining depth increased to 305 m (1,000 ft).[18] Tailgate conditions were generally good throughout the extraction of the 8th Left panel and the first half of the 9th Left panel; however, when the 9th Left face drew even with the 8th Left panel gob, conditions in the tailgate (8th Left gate road) rapidly worsened and the face advance rate was slowed to install additional secondary support. Poor tailgate ground conditions nevertheless persisted throughout the remainder of the panel, with severe roof cutter often developing along the panel rib, extreme convergence (roof sag) approaching 50% of the original entry height, and gob, containing large blocks that had to be drilled and shot, flushing around the tailgate shield onto the longwall panline. Tailgate conditions throughout the 10th Left panel retreat were equally as unfavorable as those encountered in the second half of the 9th Left longwall.[19]

CSCC began mining the ninth (most recent) panel, 11th Left, in October 1993.[20] The panel was 183 m (600 ft) wide and approximately 2,440 m (8,000 ft) long. Mining depths above the panel ranged from 275 to 335 m (900 to 1,100 ft), averaging about 305 m (1,000 ft). In an effort to further improve gate road stability, chain pillar width for the 11th Left headgate was increased to 30.5 m (100 ft) to ensure that the pillars could withstand full longwall abutment loading with minimum pillar compression and consequent roof deflection, which could lead to failure of the weak roof. Adverse tailgate ground conditions even more severe than those encountered in the preceding panels were experienced in the 11th Left panel.[21] Additionally, headgate support difficulties in an area of highly fractured, slickensided roof extending nearly 168 m (550 ft) along the 11th Left gate road significantly slowed longwall retreat during the first several months of panel extraction. Extraction of the 11th Left panel was completed in December 1994.[22]

[11]Work cited in footnote 7.

[12]Private communications from C. L. Stewart and M. A. Stevenson, Cyprus Shoshone Coal Corp., 1994.

[13]Source cited in footnote 12.

[14]Work cited in footnote 7.

[15]Work cited in footnotes 7 and 9.

[16]Source cited in footnote 12.

[17]Work cited in footnote 7.

[18]Work cited in footnote 9.

[19]Work cited in footnote 7.

[20]Source cited in footnote 12.

[21]Work cited in footnote 7.

[22]Source cited in footnote 12.

Because of the positions and orientations of the 11th Left gate road, the Barrel Springs Fault and the property boundary, as well as economic concerns regarding the large extent of main and gate road entries required to develop an adjacent tenth panel, CSCC has determined to not develop a 12th Left gate road or longwall panel in the area between the 11th Left gate road and the fault.[23] Extraction of the tenth panel is currently underway in the mine area southeast of the main entries.

GATE-ENTRY GROUND CONTROL

As mining has progressed downdip beneath increasing cover, roof control has become the most serious problem faced by CSCC. In the early panels under shallow cover, neither postdevelopment overburden loading nor longwall abutment loading were sufficient to contribute to significant ground support difficulties. Moreover, because much of the 1st to 5th Left gate roads and longwall panels were overlain by an old strip pit in the Hanna No. 82 Coalbed, only about 91 m (300 ft) of intact undisturbed rock remained above this portion of the mine, and varying depths of uncompacted spoil fill made up the bulk of the overburden in this area. Under deeper cover, however, longwall abutment loads became sufficient to seriously affect roof stability. Roof falls and entry closure in tailgate entries, such as the 9th Left gate road during 10th Left longwall panel mining, were sometimes severe enough to block the tailgate escapeway and seriously restrict ventilation airflow across the longwall face, thus occasionally necessitating temporary evacuation of the panel, with consequent production shutdowns and losses (*18*).

Gate-Entry Roof Conditions

The predominant cause of most of the Shoshone No. 1 Mine ground control problems is the extremely weak nature of the mudstone and interbedded mudstone-siltstone roof, which has the self-spanning and beam-building characteristics of unconsolidated fill (*9*). Drill-hole data and observations in high domed roof falls have indicated that 10 to 20 m (33 to 65 ft), averaging about 15.2 m (50 ft) (*9*), of this weak strata lie between the top of the coal seam and the lowest structurally competent roof sandstone.[24]

The middle 3 m (10 ft) of the 4.3-m (14-ft) seam is mined during entry development, and the face crews attempt to leave a minimum of 0.6 m (2 ft) of top coal (*10*) and sufficient bottom coal to avoid cutting into the soft, moisture-sensitive floor mudstone. Because the coal has nearly twice the compressive strength of the mudstone (according to 1992 USBM physical properties testing data), roof stability is directly dependent on the actual thickness and integrity of the top coal (*9, 10*). Longwall gate road entries are driven along the strike with a level roof and floor, as shown in figure 7; therefore, top coal thickness varies from 0.6 m (2 ft) at the downdip ("low") side of the entry to 1.5 m (5 ft) at the updip ("high") side (*10*). A USBM Coal Mine Roof Rating (CMRR) (*19*) study conducted at the Shoshone No. 1 Mine in 1993 indicated that the average CMRR (across the entry) for entries along the strike increased from 30 with no top coal at the downdip roof corner to 39 when 0.9 m (3 ft) of top coal was left at the downdip side (*10*). The relative instability of the roof at the downdip side of the entry is demonstrated by the apparent "cutter roof" condition that develops near the downdip ribline along the tailgate entries, particularly in second-panel longwall mining (*9, 18*). A finite-element modeling study (performed by geotechnical consultants retained by CSCC) showed the mudstone to yield under applied longwall abutment loading, with maximum strain at the tailgate-entry roof occurring 1.8 to 2.4 m (6 to 8 ft) from the downdip ribline (*9*). Potential shear forces (attributable to the steep dip) at the coal-mudstone interface in the immediate roof would significantly affect the stability and support capability of any support devices (such as roof bolts) emplaced into the roof. A 1993 USBM study of roof bolt loading in the 11th Left gate road showed that significantly higher bolt loadings occurred on the downdip side of the entry and that all of the downdip bolts yielded over some portion of their length (usually near the coal-mudstone interface) immediately after entry development (*4*).

Primary Roof Support

The most common roof support used throughout the Shoshone No. 1 Mine has been 1.8-m (6-ft) long, 19-mm (0.75-in) diameter, fully resin-grouted "rebar" steel roof bolts, as shown in figure 7. Additionally, following several large roof falls in the main entries in 1985, roof trusses have been installed in all entries during development. Currently, a roof truss is installed between each row of resin-grouted bolts (*10*) (figure 7).

Secondary Gate Road Support

In efforts to maintain open tailgate entries during second-panel mining, CSCC has employed several types of secondary support. In the first six panels (1st to 5th and 8th Left), the row of support shields along the face was extended with two shields in the tailgate entry. Additional support in the tailgates of the 2nd to 5th and 8th Left panels was provided by hydraulic props and timbers or yielding steel legs (used in the 4th, 5th, and 8th Left panel

[23]Source cited in footnote 12.

[24]Source cited in footnote 12.

tailgates when the untreated timbers proved susceptible to wet rot at their bases) placed in advance of the face during panel retreat.[25]

The tailgate support initially employed in the 9th Left panel was the same as that used in the preceding panels. However, when ground conditions worsened during the second half of panel retreat, CSCC began installation of four-point wood cribs in advance of the face in the tailgate entry and both inby from the face to the starting room and at least 61 m (200 ft) in advance of the face in the headgate entry. Double-row, four-point cribs were also installed in advance of the face in both the tailgate and headgate entries of the 10th Left panel throughout panel retreat, but proved incapable of supporting excessive tailgate loading under deeper cover.[26]

Secondary support in the tailgate entry of the 11th Left panel was upgraded to a two-row cribbing system with nine-point cribs on the updip (high) side and confined-core cribs (also known as "supercribs"—nine-point cribs with wooden uprights placed in the spaces between the crib blocks) (*9*) (figure 8) on the downdip (low) side. Steel beams with yielding steel legs (figure 9) and concrete "donut" cribs (figure 10) were also used on an experimental basis.[27] The standard cribs often crushed or rolled out, and in the confined-core cribs, the wooden uprights (either full-length timbers or 0.9-m (3-ft) long, 20- by 20-cm (8- by 8-in) wooden blocks placed end to end in the vertical openings in the crib) tended to buckle at the midheight of the timber or where the blocks abutted each other, then displaced the crib blocks at that point. The concrete donut cribs tended to either punch into the roof and/or floor or fail in compression and split vertically (figure 11). The yielding steel legs or posts often underwent considerable loading without yielding or showing any deflection, then suddenly and violently "yielded"—instantaneously deflecting several inches and falling away from the roof, thus losing any support capability.[28]

As a result of these problems, CSCC adopted a crib-strengthening procedure in which pairs of cribs in the downdip row (nearest the low side rib) were "wrapped" roof to floor with brattice material reinforced with 2.5- by 15-cm (1- by 6-in) boards (figure 12) and the enclosed space was filled with a low compressive strength [0.7 to 1.4 MPa (100 to 200 psi)] foamed concrete originally developed for constructing permanent ventilation seals (*20*). Because the face area of the tailgate was inaccessible to mobile haulage equipment, the foamed concrete was mixed at the surface and pumped into the tailgate entry through boreholes (drilled from the surface). The resulting "composite" cribs were approximately 3.7 m (12 ft) long and extended from the downdip (low side) panel rib to the tailgate walkway, leaving a 2.4-m (4-ft) access space between the cribs every 3.7 m along the length of the tailgate (figure 12). This system successfully controlled the tailgate ground control problems. Mine personnel feel that this effective, although costly, secondary support method distributed the support load over a larger area of the roof and alleviated the related problems of (1) overly stiff supports punching into the roof and/or floor, and (2) roof falls between the cribs. Although convergence remained between 40% and 50% of the original entry height, the composite cribs provided sufficient support to maintain a tailgate escapeway, and gob flushing around the last shield at the tailgate, a significant operational delay, was eliminated. Remote placement of foamed concrete for these cribs allowed the 11th Left panel to be extracted with record-setting production under the most difficult conditions experienced at the Shoshone No. 1 Mine.[29]

USBM GROUND PRESSURE STUDY

In 1992, the USBM began a study of ground movement and stability at the Shoshone No. 1 Mine. In early 1993, an array of instruments, including multiple-point borehole extensometers, biaxial stressmeters, and load-measuring roof bolts, was installed in a site located in the 11th Left gate road (figure 5), which was still being developed. In conjunction with this effort, hydraulic borehole pressure cells (BPC's) were installed at the site to monitor ground pressure changes during extraction of the planned 11th Left longwall panel.

BPC INSTALLATION

Ten BPC's oriented to measure changes in vertical pressure were installed from January 30 to February 3, 1993. As shown in figure 13, seven BPC's were emplaced in three boreholes drilled from the intake entry into a 30.5-m (100-ft) wide by 61-m (200-ft) long chain pillar, and three BPC's were placed in a single borehole, also drilled from the intake entry, in the solid coal block opposite the chain pillar. To maintain borehole alignment parallel to the local seam pitch, the boreholes in the chain pillar were

[25]Work cited in footnote 7.
[26]Work cited in footnote 7.
[27]Work cited in footnote 7.
[28]Work cited in footnote 12.
[29]Work cited in footnote 7.

drilled 16° above the horizontal, and the borehole in the opposite rib was drilled at the same angle below horizontal. BPC's A10, B20, C35, F45, D51, G80, and H90 were emplaced at distances of, respectively, 3 m (10 ft), 6.1 m (20 ft), 10.7 m (35 ft), 13.7 m (45 ft), 15.5 m (51 ft), 24.4 m (80 ft), and 27.4 m (90 ft) into the chain pillar from the intake entry rib, and BPC's P10, P20, and P35 were placed at respective depths of 3 m (10 ft), 6.1 m (20 ft), and 10.7 m (35 ft) into the borehole opposite the chain pillar (figure 13).

The BPC's were all pressurized to a calibrated gauge pressure of 8.62 MPa (1,250 psi), the approximate vertical pressure from overburden loading at the 335-m (1,100-ft) mining depth at the site [assuming 25 kPa vertical stress per meter of depth (1.1 psi/ft)], and the initial pressure indicated for each BPC by its respective pressure chart recorder was noted.

The hydraulic tubing connecting BPC's P10, P20, and P35 to the pressure recorders was severed by mining equipment during February 1993. During March 1993, the tubing lines were reconnected with high-pressure compression fittings, the BPC's were repressurized to the initial setting pressure, and a steel channel cover plate was installed over the tubing lines to protect them from further damage.

PRESSURE MONITORING

Because USBM longwall gate road studies in other Western U.S. coal mines (*21-23*) indicated that the onset of pressure increase attributable to front abutment loading commonly occurred about 152 to 122 m (500 to 400 ft) in advance of an approaching longwall face, continuous pressure monitoring was initiated on March 3, 1994, when the 11th Left panel face was approximately 146 m (480 ft) inby the BPC site. At this time, all BPC's except A10 indicated pressures within ±1.38 MPa (±200 psi) from the initial setting pressures. A hydraulic fitting in the pressure recorder for BPC A10 had been broken, precluding further data collection from the BPC. Also, the channel steel protecting the tubing lines for BPC's P10, P20, and P35 had been torn from the rib, exposing the lines to further damage or severing.

Either the tubing line for BPC P35 was damaged or the BPC failed on March 19, when the longwall face was approximately 127 m (416 ft) inby the BPC location, causing the recorded pressure to drop to a 1.72-MPa (250-psi) residual level, where it remained until site monitoring was terminated. The tubing lines for BPC's P10 and P20 were severed by mobile equipment on March 23, when the face was about 112 m (367 ft) inby the BPC location, precluding further pressure monitoring of the coal block opposite the chain pillar. The remaining six BPC's in the chain pillar functioned normally until monitoring was terminated.

A permanent cross seal to inhibit spontaneous combustion of remnant coal in longwall gob areas was constructed across the intake entry of the 11th Left gate road at least every three crosscuts throughout panel retreat. To prevent isolation (with consequent abandonment) of the pressure chart recorders by the seal being erected at the next chain pillar outby the site, the recorders were disconnected from the tubing lines on April 16, 1994, terminating monitoring of the remaining BPC's (B20, C35, F45, D51, G80, and H90) when the face was approximately 25 m (83 ft) (average) outby the site.

PRESSURE DATA ANALYSIS

Loading Chronology

Plots of BPC pressure compared to relative face position are shown in figures 14 and 15. In these illustrations, and throughout this report, a negative (-) relative face position indicates that the longwall face was inby a BPC (or group of BPC's in the same borehole), and a positive face position indicates that the face had progressed outby the BPC(s).

In the site pillar (figure 14), recorded pressures for all six BPC's were somewhat erratic from the beginning of monitoring at face position -146 m (-480 ft) to face position -98 m (-320 ft), at which point pressures stabilized and rose steadily until the face was about 49 m (160 ft) inby the site. The greatest degree of pressure fluctuation occurred from -146 to about -122 m (-480 to -400 ft). Interestingly, during this period, the panel face was progressing through an area of extremely poor roof conditions in the 11th Left headgate entry; consequently, the face advanced only about 1.4 m (4.7 ft) during each day of this period. When the face progressed into more favorable headgate roof conditions, the face advanced approximately 5 m (16.4 ft) during each day and pressure fluctuation decreased. These fluctuations could possibly be attributable to effects on the pressure recording mechanisms from vibrations caused by a winch or mobile equipment traffic in the same crosscut as the pressure chart recorders during the extended period of slow face progression.

Indicated pressures for the BPC's nearer the ribs of the pillar (B20, G80, and H90) were higher than pressures shown by those near the center of the pillar (C35, F45, and D51) throughout the period from the beginning of monitoring until face position -49 m (-160 ft), and the pressure increase rates for the BPC's nearest to the active panel (G80 and H90) were greater than pressure increase rates for the other BPC's during the face distance interval from -98 to -49 m (-320 to -160 ft). As the face progressed from -49 to -27 m (-160 to -90 ft), the three BPC's on the active-panel side of the pillar (H90, G80, and D51) showed significant increases in both indicated pressure and pressure increase rate, whereas the BPC's on the opposite side of the pillar (B20 and C35) both indicated an approximately 0.7-MPa (100-psi) pressure decrease. During the next 3 m (10 ft) of face retreat, BPC's F45, G80, and H90, all located in the same borehole, experienced pressure decreases of 0.7 MPa (100 psi) for BPC's F45 and G80 and 1.4 MPa (200 psi) for BPC H90. These relatively slight recorded decreases may have been caused by fracturing in the pillar near their common borehole, or may be attributable to irregularities in pressure data recording. Indicated pressure for H90 continued to decrease to about 8.3 MPa (1,200 psi) at face position -6 m (-20 ft), then increased to a maximum of 11.0 MPa (1,600 psi) at face position +6 m (20 ft outby the BPC), at which point the coal at the BPC location apparently yielded, as indicated pressure then steadily dropped to a low of 3.5 MPa (500 psi) at face position 30 m (100 ft), where monitoring ceased. Except for a minor [0.7 MPa (100 psi)] fluctuation from face position -24 to -15 m (-80 to -50 ft), pressure indicated by BPC G80 rose steadily from 10.7 MPa (1,550 psi) to 13.4 MPa (1,950 psi) as the face advanced from -24 to +15 m (-80 to +50 ft). At this point, the coal near BPC G80 [6.1 m (20 ft) from the pillar rib nearest the active panel] apparently yielded, and pressure indicated by the BPC decreased to 12.4 MPa (1,800 psi) at monitoring termination. Pressures indicated by the BPC's near the center of the pillar, D51 and F45, steadily increased from face position -24 m (-80 ft) until monitoring ceased, showing increases of 2.2 MPa (325 psi) for BPC D51 and 1.4 MPa (200 psi) for BPC F45. Pressures for the BPC's farthest from the active panel, C35 and B20, remained nearly constant from face position -27 m (-90 ft) until the face was about 3 m (10 ft) outby the site, then climbed sharply, rising 2.1 MPa (300 psi) in 15 m (50 ft) of face advance, until monitoring was terminated.

The loading chronology for BPC's P10, P20, and P35, located in the solid coal block at the opposite side of the pillar from the active panel (figure 13), is shown in figure 15. During the brief period from monitoring inception until the hydraulic tubing for these BPC's was severed, P20 and P35, located at depths of, respectively, 6.1 m (20 ft) and 10.7 m (35 ft) from the intake entry rib, both showed a gradual slight [0.7 MPa (100 psi)] pressure increase attributable to the approach of the 11th Left panel face, with the onset of the front abutment load apparently occurring between face positions -145 and -140 m (-476 and -460 ft). BPC P10, located 3 m (10 ft) from the rib, never indicated any pressure response clearly attributable to abutment loading. The pressure fluctuations shown by BPC P35 from face positions -132 to -129 m (-434 to -424 ft) and by BPC's P10 and P20 from -127 to -119 m (-416 to -392 ft) generally correspond to those of the pillar BPC's at similar face positions during the period of slow face progression.

Pillar Load Configuration Response to First-Panel Mining

Figure 16 shows profiles of BPC pressures across the study site chain pillar for six face positions during the progression of the 11th Left longwall. At face position -131.1 m (-430 ft), the pressure profile had probably changed very little from the initial configuration shortly following development of the pillar, with the highest pressures occurring at about 6.1 m (20 ft) into the pillar from the ribs. As the longwall progressed to face position -95.7 m (-314 ft), pressures on the side of the pillar nearest the longwall panel remained essentially unchanged, while the BPC's on the side opposite the panel showed a slight pressure increase. At face position -39.7 m (-130 ft), loading across the pillar had uniformly increased, maintaining essentially the same basic load configuration for the pillar. When the face had progressed to 4 m (13 ft) outby the site, loads on the side of the pillar nearest to the active panel had increased significantly, with BPC G80, 6.1 m (20 ft) from the active-panel-side rib, indicating the greatest increase. This load configuration remained essentially the same at face position 6.7 m (22 ft); however, by the time the face had progressed to 18.3 m (60 ft) outby the site, the coal near BPC H90, 3 m (10 ft) from the active-panel-side rib, had apparently yielded and a portion of the load had shifted to the opposite side of the pillar, with BPC's C35 and B20 showing significant pressure increases.

The load configuration across the pillar indicated by the initial BPC pressure profile at face position -131.1 m (-430 ft) generally conforms to the "confined-core" pillar loading geometry (figure 17) postulated by A. H. Wilson, noted British ground control authority (*24-25*). For a yielding roof and floor (definitely the Shoshone mining environment), Wilson's equations are:

$$X_b = \frac{M}{2}\left[\left(\frac{q}{p + p'}\right)^{\frac{1}{k-1}} - 1\right], \tag{1}$$

$$\sigma_{max} = kq + \sigma_0, \tag{2}$$

and

$$k = \frac{1 + \sin\phi}{1 - \sin\phi}, \tag{3}$$

where X_b = distance from pillar rib to point of maximum pillar stress,

M = mining height,

q = overburden stress (overburden density multiplied by depth),

p = support resistance against rib,

p' = cohesion strength of broken coal,

k = triaxial stress factor,

σ_{max} = maximum pillar stress,

σ_0 = in-situ uniaxial compressive strength (approximately 1/5 laboratory uniaxial compressive strength),

ϕ = angle of internal friction of coal seam.

For the 11th Left gate road site, M = 3.05 m (10 ft), q = 8.34 MPa (1,210 psi), p = 0, p′ = 0.096 MPa (13.89 psi) [from Wilson (*24-25*)], k = 4.815, σ_0 = 3.5 MPa (514 psi) (from table 1), and ϕ = 41° (from table 1). Therefore X_b = 3.4 m (11.1 ft) and σ_{max} = 43.7 MPa (6,340 psi). The disparity between the magnitude of σ_{max} and X_b [43.7 MPa (6,340 psi) at 3.4 m (11.1 ft) into the pillar from the ribs] indicated by the Wilson equations and those shown by the initial BPC pressure profile [9 MPa (1,300 psi) at 6.1 m (20 ft) from the rib] may be attributed to several factors. The pressure profiles in figure 16 are biased according to the BPC locations and cannot reflect loading between adjacent BPC's or between the outside BPC's (H90 and B20) and the pillar ribs. The location of the maximum pillar stress may have been between BPC's H90 and G80, in which case BPC H90 would have been in the yield zone (figure 17) and BPC G80 would have been in the elastic "confined core." Also, the Wilson equations are extremely sensitive to the value of ϕ used to calculate k. For example, if ϕ = 30° [a more "common" value for ϕ (*25*)] is used in the Wilson calculations for the Shoshone 11th Left site, k = 3 and σ_{max} = 28.6 MPa (4,144 psi) and is located at a distance X_b of 12.7 m (42.7 ft) from the rib(s). According to Babcock's analysis of BPC reaction to applied load with respect to the physical properties of the "host" rock (*26*), the BPC data may not reflect actual vertical stresses across the pillar, owing to BPC load response effects caused by the dissimilar relative stiffnesses of the "rigid" concrete-encapsulated steel BPC's and the "soft" Shoshone No. 1 Mine coal.

The higher indicated pressures on the side of the pillar nearest the retreating 11th Left longwall panel throughout the monitoring period conform to similar pillar load responses to longwall mining reported by numerous observers at various mine locations across the United States (*21-23, 27-29*).

The BPC data were generally corroborated by the data from biaxial stressmeters emplaced in the study site pillar as part of the concurrent USBM strata movement and deformation study.[30] The relatively low magnitude of the BPC pressures at the Shoshone No. 1 Mine site, compared to those measured in other Western U.S. longwall gate roads (both two- and three-entry) (*21-23*) may be attributable to the relative "softness" of the Shoshone immediate roof, coal seam, and mine floor contrasted to those of the other gate road study sites.

GATE ROAD STABILITY ANALYSIS FOR SHOSHONE NO. 1 MINE

The structural stability of the Shoshone No. 1 Mine gate roads was evaluated using the Analysis of Longwall Pillar Stability (ALPS) method developed by the USBM (*30-31*). The ALPS method provides a practical, convenient means of sizing or evaluating the stability of abutment-type chain pillars (pillars designed to withstand all loading during the entire course of longwall mining) in two-entry and multientry gate road systems. The ALPS method was chosen for the Shoshone gate roads analysis in preference to other well-known, commonly used longwall gate pillar design methods because of its adaptability to the various gate road designs used at the mine and its particular applicability to U.S. longwall mining conditions and design practices (owing to its basis on a large number of actual U.S. longwall case histories and engineering field studies) (*30-31*). The pillar sizing methods developed by Hsiung and Peng (*32*) and Choi and McCain (*33*) are intrinsically limited to three-entry gate road systems, using equal-sized chain pillars in the Hsiung and Peng method, and a row of abutment pillars paired with a second row of 9.75-m (32-ft) yield pillars in the Choi and McCain approach. In contrast, of the 11 gate roads at the Shoshone No. 1 Mine, 6 were developed using two-entry designs with abutment pillars. Although the Carr and Wilson pillar design approach (*25*) is applicable to both two-entry and multientry gate road designs, it is highly sensitive to the values used for the angle of internal friction (ϕ), cohesion strength (p'), and in-situ uniaxial compressive strength (σ_0) of the coal in the Wilson equations (*24-25*) and may not be applicable to the unique physical properties of the Shoshone No. 1 Mine coal. Also, the Carr and Wilson method was devised to account for gate road performance observed in the ground conditions generally found in the coalfields of the United Kingdom and northern Alabama and may not apply to the

[30]Partial Analysis of Biaxial Stressmeter Data. Memorandum from M. K. Larson, USBM, 1994, 7 pp.

unique conditions of the Shoshone No. 1 Mine—extremely weak roof, weak floor, and soft coal.

The ALPS pillar design approach evaluates gate road chain pillar stability at five stages of first- and second-panel longwall mining, each having a characteristic load configuration, as shown in figure 18: A, development loading, which is the load on the pillars before longwall mining and equals the tributary area load; B, headgate loading, the load at the headgate corner of the first-panel face, which is the first-panel front abutment load plus the development load; C, bleeder loading, the load on a gate road system adjacent to a mined-out (first) panel, which equals the full first-panel side abutment load plus the development load; D, tailgate loading, the load at the tailgate corner of the second panel, which equals the front abutment load of the second panel plus the side abutment load of the first panel plus the development load; and E, isolated loading, which is the loading on a gate road isolated between two mined-out panels and equals the full side abutment load from both panels. Gate road stability for isolated loading is usually of little concern, since the gate road will no longer be used to access any active panel and is commonly inaccessible. For each longwall mining stage and load configuration, the load-bearing capacity of the chain pillars is divided by the load applied on the gate road to obtain the appropriate stability factor. Because most gate-entry systems are used twice, first as a headgate and then as a tailgate, tailgate loading stability is used as the critical design criterion and is emphasized in the ALPS evaluation. This emphasis is definitely appropriate for the Shoshone No. 1 Mine, where the most adverse ground conditions, particularly severe convergence and roof falls, have been encountered in the tailgate entries under deep cover during second-panel mining.

ALPS EVALUATION OF 11TH LEFT GATE ROAD

The following input parameters were used in an ALPS evaluation of the Shoshone 11th Left two-entry gate road: 3-m (10-ft) mining height, 335-m (1,100-ft) cover depth, 6.2-MPa (900-psi) in-situ coal strength, 21° abutment angle, 183-m (600-ft) panel width, 5.5-m (18-ft) entry and crosscut width, 61-m (200-ft) chain pillar length, and 30.5-m (100-ft) pillar width. The analysis showed the gate road to have the following stability factors: development loading (SFd) = 2.39, headgate loading (SFh) = 1.47, bleeder loading (SFb) = 1.06, tailgate loading (SFt) = 0.63, and isolated loading (SFi) = 0.56. Therefore, according to the ALPS criteria, the gate road was marginally stable after the extraction of the 11th Left longwall panel (SFb) and probably would have proven unstable if a subsequent panel had been extracted (SFt).

ALPS EVALUATION OF SHOSHONE NO. 1 MINE GATE ROADS USING CMRR SYSTEM

In further development of the ALPS design method, the USBM related gate road stability predicted by the ALPS method to gate-entry roof conditions quantified according to the CMRR (*34-36*). A study based on the overall ground support performance of 69 longwall gate road systems (located across the U.S. coalfields), compared to the ALPS stability factors and CMRR calculated for each gate road site, developed and validated the empirical relationship expressed in the simple equation:

$$\text{ALPS SFr} = 1.76 - 0.014\ \text{CMRR}, \tag{4}$$

where ALPS SFr = ALPS stability factor suggested for design.

Since tailgate loading is the most critical stage of longwall mining for Shoshone No. 1 Mine gate road stability, the equation used in the "ALPS with CMRR" gate road analysis was

$$\text{SFt} = 1.76 - 0.014\ \text{CMRR}. \tag{5}$$

The 1993 CMRR study at the mine showed that Shoshone gate entries with 6.1 m (2 ft) of top coal at the downdip (low side) rib (the minimum top coal thickness generally maintained in the gate road entries) had an average CMRR of 37 (*10*); therefore, in accordance with equation 5, a SFt of 1.24 was used as the critical stability criterion in an ALPS back analysis of the Shoshone gate roads used as tailgates in second-panel longwall mining (2nd to 5th Left and 8th to 10th Left). The results of the back analysis are shown in figure 19. Two curves, one for three-entry gate roads with equal-width pillars and the other for two-entry gate roads, were developed to establish minimum pillar widths required to meet the SFt = 1.24 criterion for the range of mining depths in which each design was utilized. The ALPS SFt for each tailgate road was calculated using the pillar dimensions, average mining depth, panel width [122 m (400 ft) for the three-entry gate roads and 183 m (600 ft) for the two-entry gate roads], and entry and crosscut width [5.5 m (18 ft) in all cases] shown in figure 5 for each gate road; a 6.2 MPa (900 psi) in-situ coal strength and 21° abutment angle were used in all calculations. Figure 19 clearly shows that the pillar widths in the three-entry gate roads under shallow cover were more than adequate to satisfy the SFt = 1.24 criterion, whereas the pillar widths in the two-entry gate roads under deep cover were insufficient to meet the criterion. Moreover, the SFt's for the shallow-cover, three-entry gate roads ranged from 117% to 276% greater than 1.24, while the SFt's of the deep-cover, two-entry gate roads ranged from 18.5% to 50% below 1.24. Since relatively few ground support problems were encountered in any of the shallow-cover, three-entry gate roads during second-panel mining, in contrast to the severe entry closure and roof problems experienced in the deep-cover, two-entry gate roads, equation 5 seems to be a valid standard for evaluating the tailgate loading stability of the Shoshone gate roads. Because the chain pillars in the two-entry gate roads were not large enough to have sufficient load-bearing capacity to withstand the loads imposed on them, particularly in second-panel mining, they may be considered as "critical pillars" (pillars too small to function as abutment pillars, but too large to function as yield pillars) (*37-38*).

ALTERNATIVES FOR IMPROVING TAILGATE LOADING STABILITY

The first alternative for improving tailgate loading stability in the Shoshone No. 1 Mine two-entry gate roads—increasing the chain pillar width—is suggested by figure 19. Other alternatives, such as decreasing gate-entry width and increasing top coal thickness, were suggested in the 1993 USBM CMRR study.[31]

Increased Chain Pillar Width

Numerous field studies of ground pressures in longwall mining have reported that the magnitude of measured side abutment loading is inversely proportional to the distance between the measuring instruments and the nearest edge of the longwall gob. Airey (*39*) hypothesized that within the area influenced by side abutment loading, the stress declines according to the inverse square of the distance from the panel edge. To characterize the side abutment loading configuration and calculate the magnitude of side abutment loads for the ALPS evaluations, Mark (*30-31*) utilized the equations

$$\sigma a = \left[\frac{3Ls}{D^3}\right](D - x)^2, \qquad (6)$$

$$Ls = H^2 (\tan\beta) \left[\frac{\tau}{2}\right], \qquad (7)$$

and

$$D = 9.3\sqrt{H}, \qquad (8)$$

[31]Work cited in footnote 9.

where σa = side abutment stress at a distance (x) from the edge of a longwall panel (abutment stress distribution function),

Ls = total side abutment load,

D = extent of side abutment influence zone,

H = mining depth,

β = longwall abutment angle,

and τ = overburden density.

In the case of a two-entry gate road (figure 20), the load on the entry at the opposite side of the chain pillars from the active panel (the entry adjacent to the 2nd panel in bleeder loading in figure 18) may be expressed as

$$Le = \int_a^b \left[\frac{3Ls}{D^3}\right](D - x)^2\, dx, \qquad (9)$$

where Le = abutment load on entry per unit entry length,

b = distance from edge of longwall gob to tailgate rib of next panel,

a = width of chain pillar,

and $b - a$ = width of entry.

By integrating equation 9 and substituting the terms for Ls and D from equations 7 and 8, the equation for calculating the full side abutment load on the remote entry becomes

$$Le = \frac{\tau(\tan\beta)\sqrt{H}}{9.3^3}\left[(9.3\sqrt{H} - a)^3 - (9.3\sqrt{H} - b)^3\right]. \qquad (10)$$

Using the 11th Left mining depth (H) = 335 m (1,100 ft), longwall abutment angle (ß) = 21° (from Mark (*30-31*), and overburden density (τ) = 2,537 kg/m³ (158.4 pcf) [from Djahanguiri (*5*)], equation 10 was utilized to develop the entry load curves shown in figure 21 for entry widths (b-a) of 4.9 m (16 ft), 5.5 m (18 ft), and 6.1 m (20 ft). As demonstrated by figure 21, substantial reductions in loading on the remote entry may be achieved by increasing the chain pillar width. For example, by increasing the width of the 11th Left chain pillars from 30.5 to 36.6 m (100 to 120 ft) with 5.5-m (18-ft) entry width, the load on the remote (intake) entry would hypothetically be reduced by 19% after extraction of the 11th Left longwall panel (ALPS bleeder loading). Figure 21 also shows that significant entry load decreases may be realized by reducing entry width. As a case in point, for a 30.5-m (100-ft) pillar width, decreasing the entry width from 5.5 to 4.9 m (18 to 16 ft) results in a 10% reduction in entry load. To realize an equivalent entry load reduction while retaining the 5.5-m (18-ft) entry width would require increasing the pillar width to approximately 33.5 m (110 ft). Equation 10 and figure 21 are intended only to calculate and illustrate ***hypothetical*** first-panel-mining side abutment loading on the tailgate entry of the second panel; they do not represent actual loads across an entry roof and ***should not*** be used for gate-entry roof support design.

Decreased Entry Width and Increased Top Coal Thickness

A second ALPS back analysis was used to determine the effects of entry width and top coal thickness on gate road stability in tailgate loading. The 1993 CMRR study reported that the Shoshone No. 1 Mine gate entries with 0.91 m (3 ft) of top coal at the low side rib had an average CMRR of 39 (*10*). Therefore, using the criterion of equation 5, the required SFt for tailgate loading stability is 1.21, as shown in figure 22. The results of the second ALPS back analysis are shown in figure 23. Two sets of curves were developed, one for SFt = 1.24 [0.61 m (2 ft) of top coal] (figure 22) and the other for SFt = 1.21, to establish minimum pillar widths required to meet these criteria for mining depths ranging from 152 to 396 m (500 to 1,300 ft) and tailgate-entry widths of 4.9 m (16 ft), 5.5 m (18 ft), and 6.1 m (20 ft). The influence of top coal thickness (and thence roof quality) on tailgate loading stability is evident in figure 23. For example, at 335-m (1,100-ft) mining depth with a 5.5-m (18-ft) entry width, the pillar width required to meet the equation 5 criterion is decreased from 46.3 to 43 m (152 to 141 ft) by increasing the top coal thickness from 0.61 to 0.91 m (2 to 3 ft). The 0.3-m (1-ft) increase in top coal thickness results in reductions in pillar widths (conforming to the equation 5 standard) ranging from 6.6% at a mining depth of 152 m (500 ft) to 7.2% at a mining depth of 396 m (1,300 ft).

CONCLUSIONS

The singularly unfavorable ground conditions at the Shoshone No. 1 Mine—an exceptionally weak immediate roof and a steeply dipping low-strength coal seam prone to spontaneous combustion—present a challenging environment for longwall mining. The first five longwall panels were developed with three-entry gate roads and were extracted with few ground control problems. To minimize the likelihood of spontaneous combustion and to improve gate road stability for increasing mining depths, two-entry gate roads with larger pillars were utilized in the next four panels. However, severe roof falls and entry closure occurred in the tailgate entry of each panel. Several primary and secondary roof support devices and systems were employed with limited success (except for the wood-foamed concrete "composite" cribs used in the tailgate of the last panel) in efforts to stabilize the tailgate entries.

The USBM pillar pressure study indicated a postdevelopment pillar load distribution similar to Wilson's (*24-25*) "confined-core pillar" concept and side abutment load response to first-panel mining comparable to that reported in other gate road pillar pressure studies. The low pressures (compared to pressures measured in the other studies) are probably attributable to the relative softness of the mine seam and adjacent strata.

An ALPS evaluation of the headgate road of the last longwall panel indicated marginal stability in first-panel mining and inadequate stability for second-panel mining. The ALPS method with the CMRR system was used to analyze second-panel-mining (ALPS tailgate loading) stability of the Shoshone No. 1 Mine second-panel-mining tailgate roads and to evaluate alternatives for improving tailgate stability for future panels. Using the ALPS SFt-CMRR relationship of equation 5 as the primary gate road stability criterion implies three basic assumptions:

- ALPS is a valid method for evaluating the stability of the gate road(s). The ALPS method was specifically devised for sizing abutment-type longwall gate road pillars and analyzing the stability of gate roads utilizing them. All Shoshone No. 1 Mine gate road pillars were designed (or intended) to function as abutment pillars.
- Roof quality and stability are the predominant ground control parameters affecting gate road stability (rather than roof-floor pillar confinement, weak heave-prone floor, highly fractured coal, etc.). Entry closure, primarily due to roof sag, roof "guttering" (cutter roof) along the tailgate panel ribs, and roof falls have been the most severe ground control problems experienced in the Shoshone No. 1 Mine.
- Tailgate loading (second-panel mining) is the most critical loading condition and phase of double-panel extraction for gate road stability. Most of the ground control problems and hazards encountered in longwall operations at the Shoshone No. 1 Mine have occurred in the tailgate entries during second-panel mining.

The ALPS SFt-CMRR evaluation of the Shoshone No. 1 Mine second-panel-mining tailgate roads, together with the history of relatively few ground control problems in the shallow-cover, three-entry gate roads compared to the severe difficulties experienced in the deep-cover, two-entry gate roads, indicated that the apparently oversized chain pillars in the three-entry gate roads were more than adequate for maintaining gate road stability in tailgate loading, whereas the apparently undersized pillars in the two-entry gate roads failed to meet this criterion.

The analysis of the effect of chain pillar width on hypothetical tailgate-entry loading from first-panel mining (in a two-entry gate road) and the ALPS-CMRR evaluation of the influence of top coal thickness and entry width on tailgate loading stability together demonstrated that increasing chain pillar width and top coal thickness and decreasing entry width are viable alternatives for improving gate road stability in second-panel longwall mining. These procedures could be carried out concurrently, limited by trailing cable restrictions and efficient cut sequencing factors during gate road development and by ventilation considerations and regulatory constraints throughout all phases of longwall mining.

ACKNOWLEDGMENTS

The authors wish to express their appreciation to Collin L. Stewart, continuous mining coordinator (former manager of technical services), and Michael A. Stevenson, former geologist, Cyprus Shoshone Coal Corp., Hanna, WY, for their assistance in BPC installation and maintenance and for providing much of the background information concerning mining conditions, experience, and procedures at the Shoshone No. 1 Mine. Particular recognition is also extended to William J. Wuest, mining engineer, USBM's, Denver Research Center, Denver, CO, for his assistance in BPC installation, and to Mark K. Larson, mining engineer, USBM's Spokane Research Center, Spokane, WA, for conducting all BPC monitoring and data collection during the monitoring period.

REFERENCES

1. Glass, G. B., and J. T. Roberts. Coals and Coal-Bearing Rocks of the Hanna Coal Field, Wyoming. WY Geol. Surv. Rep. Invest. 22, 1980, 43 pp.

2. Jackson, D. Miners Cut Through a Thick Seam. Coal Age, v. 85, No. 8, 1980, pp. 80-87.

3. Glass, G. B. Structure, Coal Thickness, and Shallow Overburden for Coal Beds No. 89, No. RME 92, No. 88, No. 86, No. 80, No. 77, No. 76, No. 73, and Hanna No. 5 in the Hanna Mining District, Hanna Coal Field, Southcentral Wyoming. WY Geol. Surv. Open File Rep. 79-5, 1979, 1 map.

4. Signer, S. P. Field Evaluations of Grouted Roof Bolts. Paper in New Technology for Longwall Ground Control. USBM Spec. Publ. 01-94, 1994, pp. 91-102.

5. Djahanguiri, F. Rock Mechanics for a Longwall Mine Design; Carbon County Coal Company. Paper in Energy Resources and Excavation Technology, Proceedings of the 18th U.S. Symposium on Rock Mechanics (Keystone, CO, June 22-24, 1977), ed. by F. D. Wang and G. B. Clark. CO Sch. Mines, Golden, CO, 1977, pp. 1C5-1 to 1C5-12.

6. Averitt, P. Coal Resources of the United States, January 1, 1974. U.S. Geol. Surv. Bull. 1412, 1975, 131 pp.

7. Weimer, R. J. Upper Cretaceous Stratigraphy, Rocky Mountain Area. Am. Assoc. Pet. Geol. Bull., v. 44, 1960, pp. 1-20.

8. Dobbin, C. E., C. F. Bowen, and H. W. Hoots. Geology and Coal and Oil Resources of the Hanna and Carbon Basins, Carbon County, Wyoming. U.S. Geol. Surv. Bull. 804, 1929, 88 pp.

9. Schissler, A. P. Cyprus Shoshone Tailgate Roof Control: Case Study. Paper in Proceedings of the 12th International Conference on Ground Control in Mining (Morgantown, WV, Aug. 3-5, 1993), ed. by S. S. Peng. WV Univ., 1993, pp. 44-46.

10. Mark, C., G. M. Molinda, A. P. Schissler, and W. J. Wuest. Evaluating Roof Control in Underground Coal Mines With the Coal Mine Roof Rating. Paper in Proceedings of the 13th International Conference on Ground Control in Mining (Morgantown, WV, Aug. 2-4, 1994), ed. by S. S. Peng. WV Univ., 1994, pp. 252-260.

11. Bieniawski, Z. T. Engineering Classification of Jointed Rock Masses. Civ. Eng. S. Afr., Dec. 1973, pp. 335-343.

12. Glass, G. B. Mining in the Hanna Coal Field. WY Geol. Surv. Misc. Rep., 1972, 45 pp.

13. Wyoming State Inspector of Mines. List of Producing Coal Mines. Annual Reports of the State Inspector of Mines of Wyoming, 1971-93. Off. State Inspector Mines, Rock Springs, WY.

14. Kalia, H. N., J. T. Schultz, and P. Wyckoff. A Ground Control Study at Vanguard No. 2 Mine, Hanna, Wyoming. Paper in Proceedings of the 20th U.S. Symposium on Rock Mechanics (Austin, TX, June 4-6, 1979). TX Univ., Austin, TX, 1979, pp. 67-74.

15. Guccione, E. (ed.). Carbon No. 1 Mine: Off to a Good Start and On to Longwall Mining. Coal Min., v. 17, No. 12, 1980, pp. 46-49.

16. Coal Age. Hanna Mine Suspends Production. V. 91, No. 6, 1986, p. 17.

17. ____. Cyprus Buys Hanna Mine. V. 92, No. 9, 1987, p. 40.

18. Moulton, C. MSHA, Cyprus Shoshone Work on Coal Mine Safety in Hanna. Star-Tribune, Casper, WY, Sept. 20, 1992, p. A6.

19. Molinda, G. M., and C. Mark. Coal Mine Roof Rating (CMRR): A Practical Rock Mass Classification for Coal Mines. USBM IC 9387, 1994, 83 pp.

20. Amick, M., J. Mazzoca, and D. Vosefski. The Use of Foamed Cement Cribs at American Electric Power Fuel Supply Meigs Division. Paper in Proceedings of the 12th International Conference on Ground Control in Mining (Morgantown, WV, Aug. 3-5, 1993), ed. by S. S. Peng. WV Univ., 1993, pp. 55-58.

21. DeMarco, M. J., J. R. Koehler, and P. H. Lu. Characterization of Chain Pillar Stability in a Deep Western Coal Mine—A Case Study. Min. Eng., v. 40, No. 12, 1988, pp. 1115-1119.

22. Barron, L. R. Longwall Stability Analysis of a Deep, Bump-Prone Western Coal Mine-Case Study. Paper in Proceedings of the 9th International Conference on Ground Control in Mining (Morgantown, WV, June 4-6, 1990), ed. by S. S. Peng. WV Univ., 1990, pp. 142-149.

23. Barron, L. R., M. J. DeMarco, and R. O. Kneisley. Longwall Gate Road Stability in Four Deep Western U.S. Coal Mines. USBM IC 9406, 1994, 84 pp.

24. Wilson, A. H. Stress and Stability in Coal Ribsides and Pillars. Paper in Proceedings of the 1st Conference on Ground Control in Mining. WV Univ., 1981, pp. 1-12.

25. Carr, F., and A. H. Wilson. A New Approach to the Design of Multi-Entry Developments for Retreat Longwall Mining. Paper in Proceedings of the 2nd Conference on Ground Control in Mining (Morgantown, WV, July 19-21, 1982), ed. by S. S. Peng and J. H. Kelly. WV Univ., 1982, pp. 1-21.

26. Babcock, C. O. Equations for the Analysis of Borehole Pressure Cell Data. Paper in Rock Mechanics: Key to Energy Production, Proceedings of the 27th U.S. Symposium on Rock Mechanics (AL Univ., Tuscaloosa, AL, June 23-25, 1986), ed. by H. H. Hartman. Soc. Min. Eng. AIME, Littleton, CO, 1986, pp. 233-240.

27. Allwes, R. A., J. M. Listak, G. J. Chekan, and D. R. Babich. The Effects of a Retreating Longwall on a Three-Entry Gate Road System. USBM RI 8996, 1985, 19 pp.

28. Heasley, K. A., and K. Barron. A Case Study of Gate Pillar Response to Longwall Mining in Bump-Prone Strata. Paper in Longwall USA Conference Papers (Pittsburgh, PA, Sept. 1988). Maclean Hunter, 1988, pp. 92-105.

29. Mark, C. Field Measurements of Chain Pillar Response to Longwall Abutment Loads. Paper in Proceedings of the 5th Conference on Ground Control in Mining (Morgantown, WV, 1986), ed. by S. S. Peng. WV Univ., 1986, pp. 114-122.

30. ____. Pillar Design Methods for Longwall Mining. USBM IC 9247, 1990, 53 pp.

31. Mark, C. Analysis of Longwall Pillar Stability (ALPS): An Update. Paper in Proceedings of the Workshop on Coal Pillar Mechanics and Design. USBM IC 9315, 1992, pp. 238-249.

32. Hsiung, S. M., and S. S. Peng. Chain Pillar Design for U.S. Longwall Panels. Min. Sci. Technol., v. 2, 1985, pp. 279-305.

33. Choi, D. S., and D. L. McCain. Design of Longwall Systems. Trans. Soc. Min. Eng. AIME, v. 268, 1980, pp. 1761-1764.

34. Mark, C., and F. E. Chase. Gate Entry Design for Longwalls Using the Coal Mine Roof Rating. Paper in Proceedings of the 12th International Conference on Ground Control in Mining (Morgantown, WV, Aug. 3-5, 1993), ed. by S. S. Peng. WV Univ., 1993, pp. 76-83.

35. Mark, C., and G. Molinda. CMRR Aids in Longwall Gate Entries Design. COAL Mag., v. 99, No. 2., 1994, pp. 40-42.

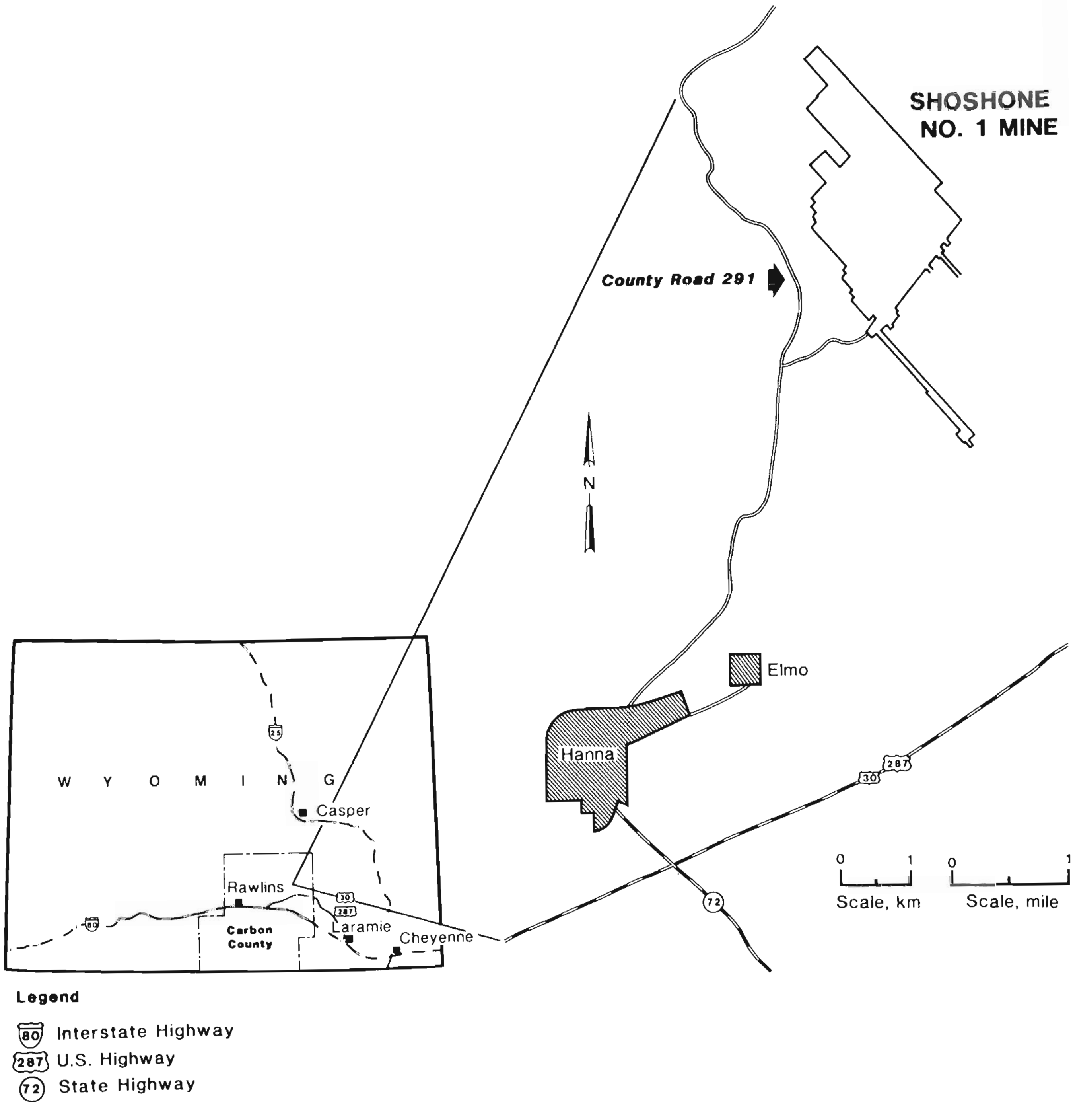
36. Mark, C., F. E. Chase, and G. M. Molinda. Design of Longwall Gate Entry Systems Using Roof Classification. Paper in New Technology for Longwall Ground Control. USBM Spec. Publ. 01-94, 1994, pp. 5-17.

37. DeMarco, M. J. Yielding Pillar Gate Road Design Considerations for Longwall Mining. Paper in New Technology for Longwall Ground Control. USBM Spec. Publ. 01-94, 1994, pp. 19-36.

38. Koehler, J. R., M. J. DeMarco, and W. J. Wuest. The Critical Pillar Concept in Yield Pillar-Based Longwall Gate Road Design (Pres. at SME Annu. Meet., Denver, CO, Mar. 6-9, 1995). Soc. Min. Eng. AIME, Littleton, CO, preprint 95-123, 1995, 14 pp.

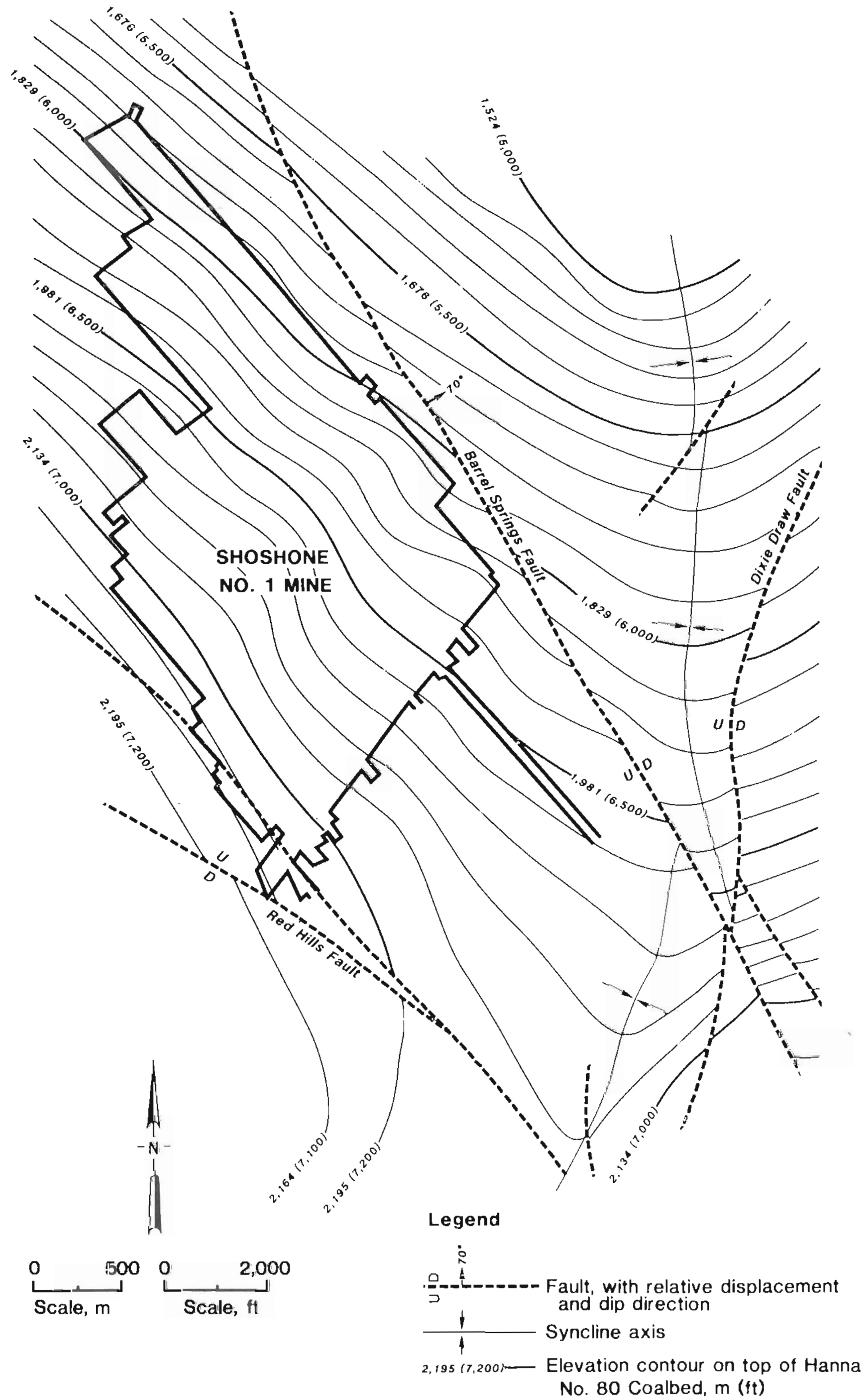
39. Airey, E. M. An Analysis of Stress Round Mine Roadways: Stresses in the Failed Rock. Appendix to Study of Yield Zones Around a Roadways. Publ. Eur. Communities, Luxembourg, Rep. EUR 5825e, 1977, pp. 44-88.

Figure 1



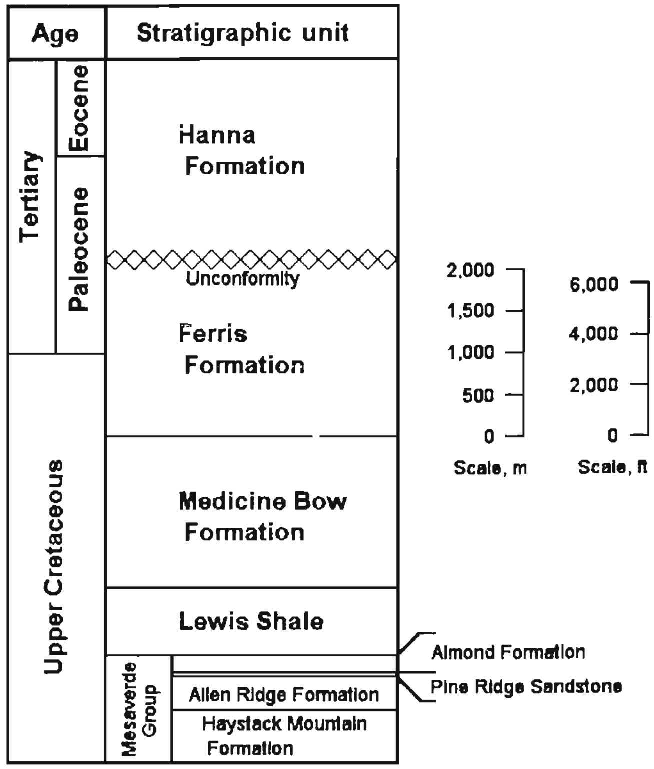
Location map for Shoshone No. 1 Mine.

Figure 2



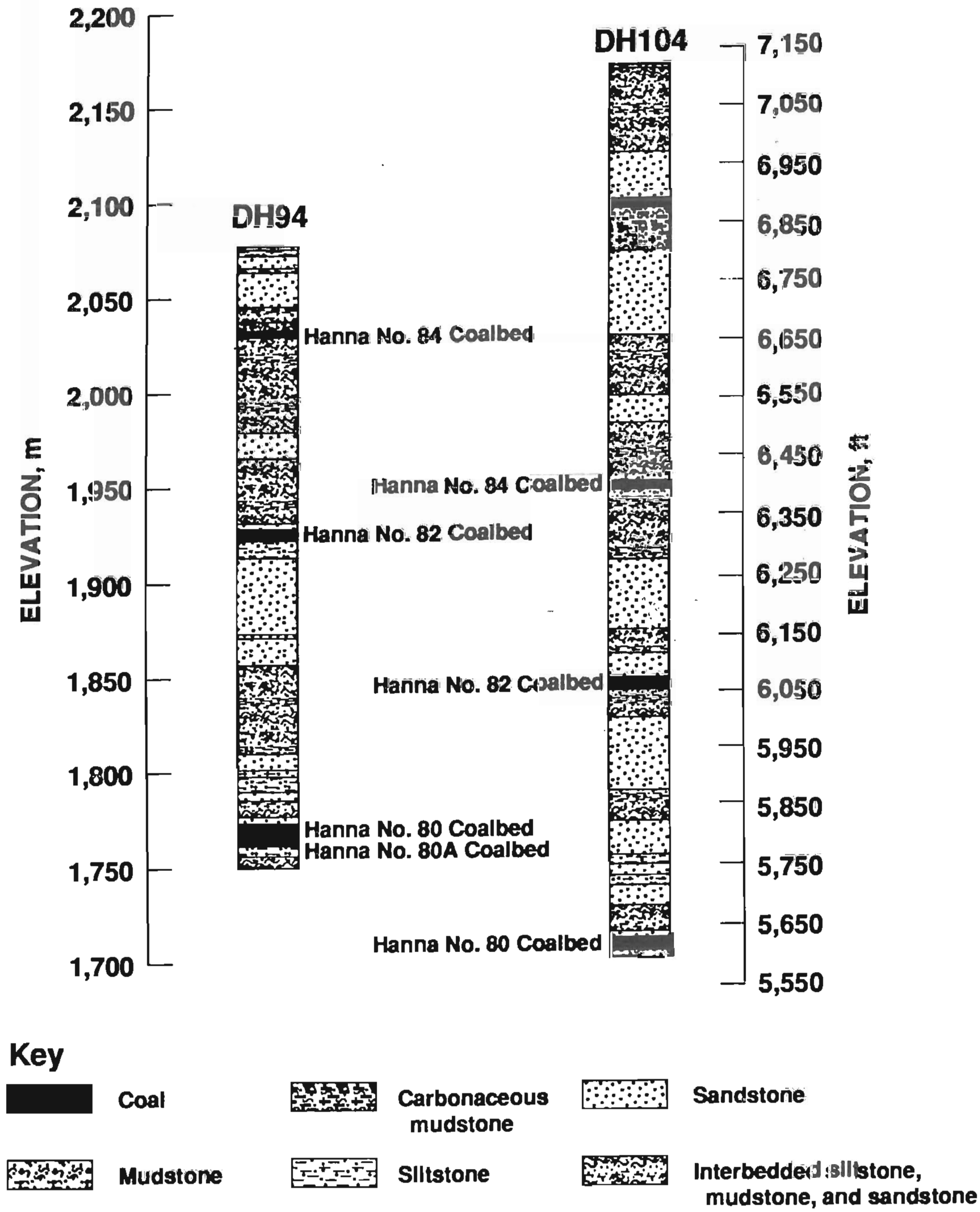
Geologic structure at Shoshone No. 1 Mine. [Modified from Glass (3)]

Figure 3



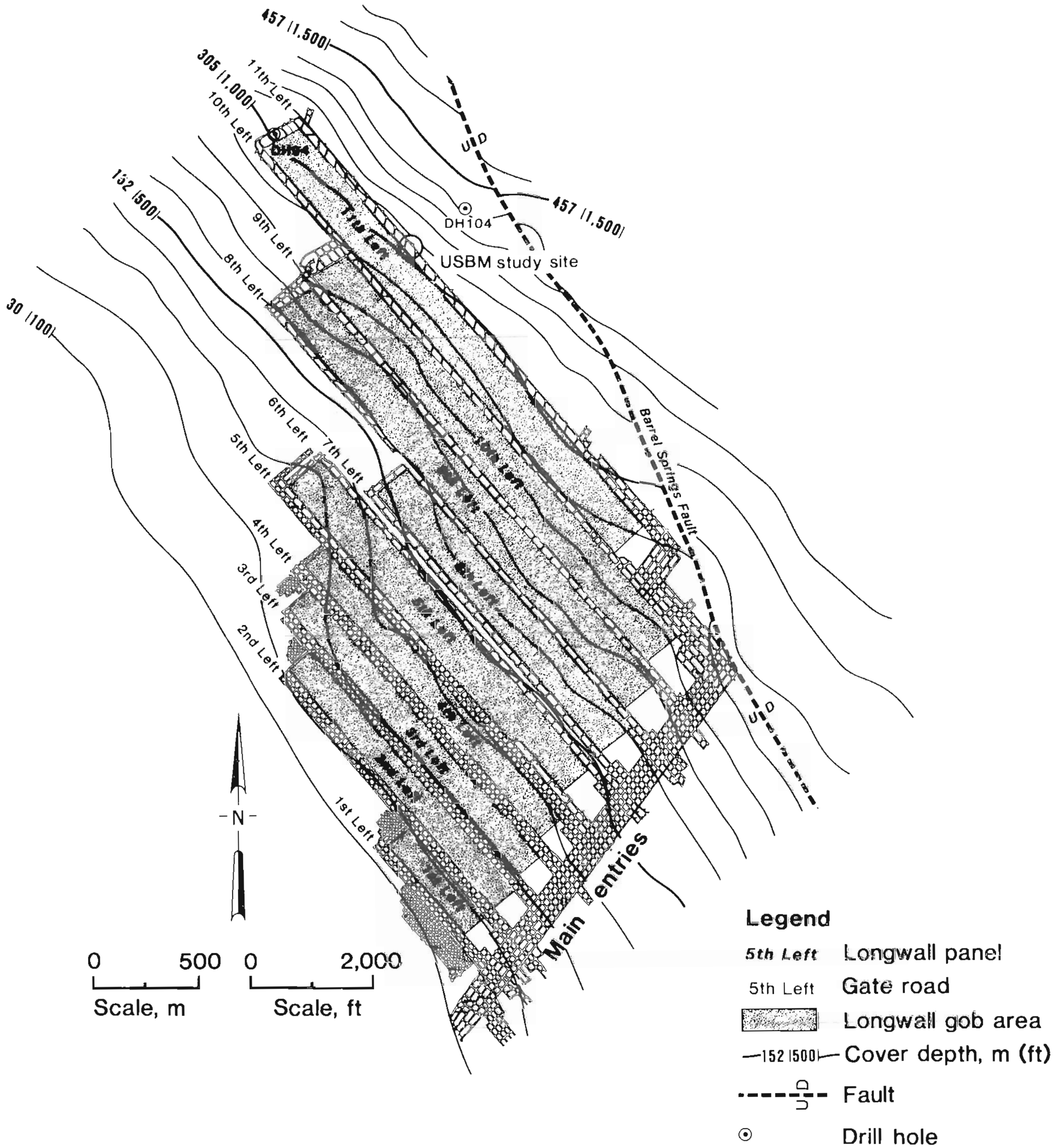
Generalized stratigraphic section for Hanna Basin. [From Glass and Roberts (1)]

Figure 4



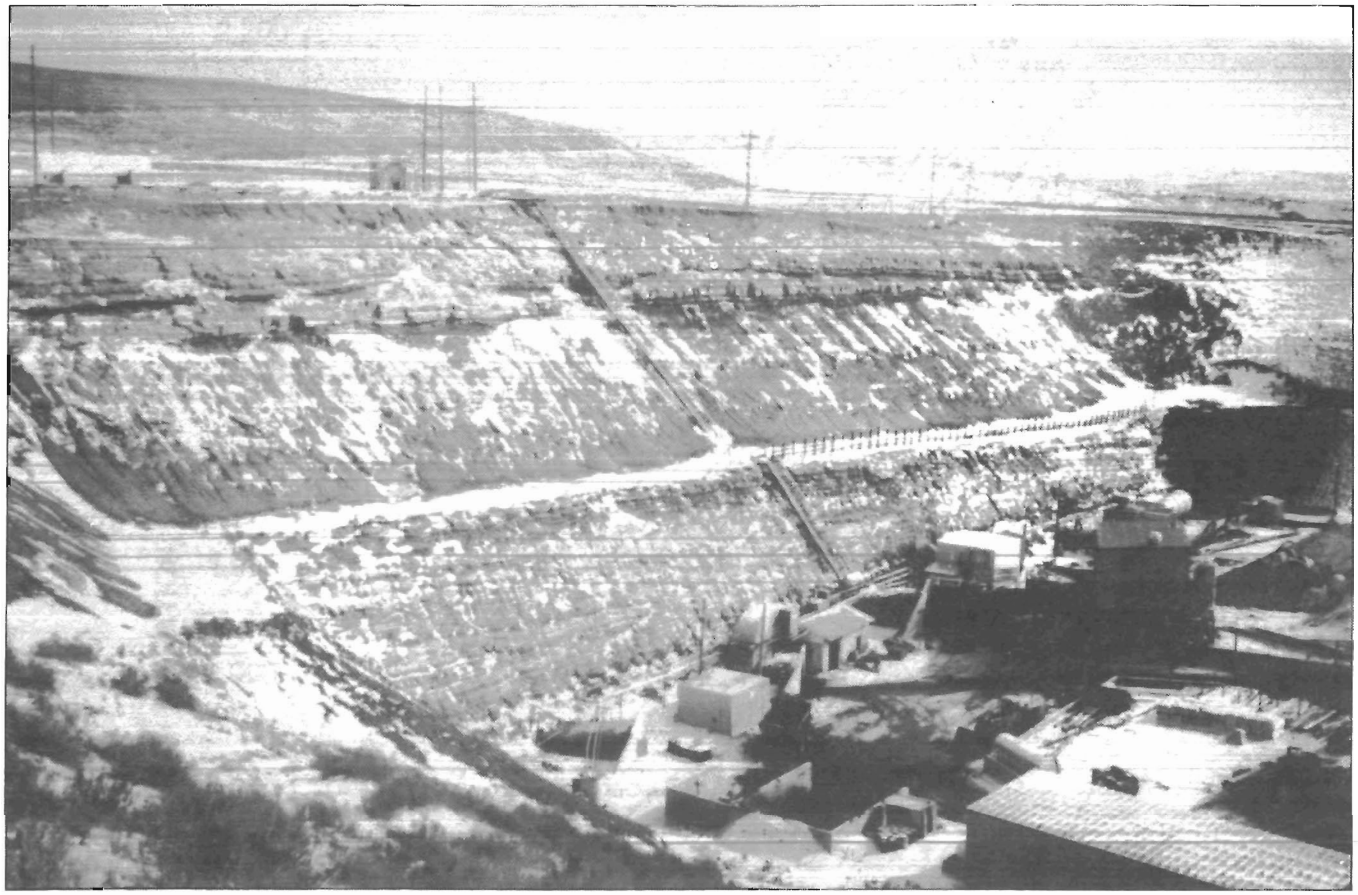
Stratigraphic columns for drill holes DH94 and DH104 (Drill-hole data courtesy Cyprus Shoshone Coal Corp.)

Figure 5



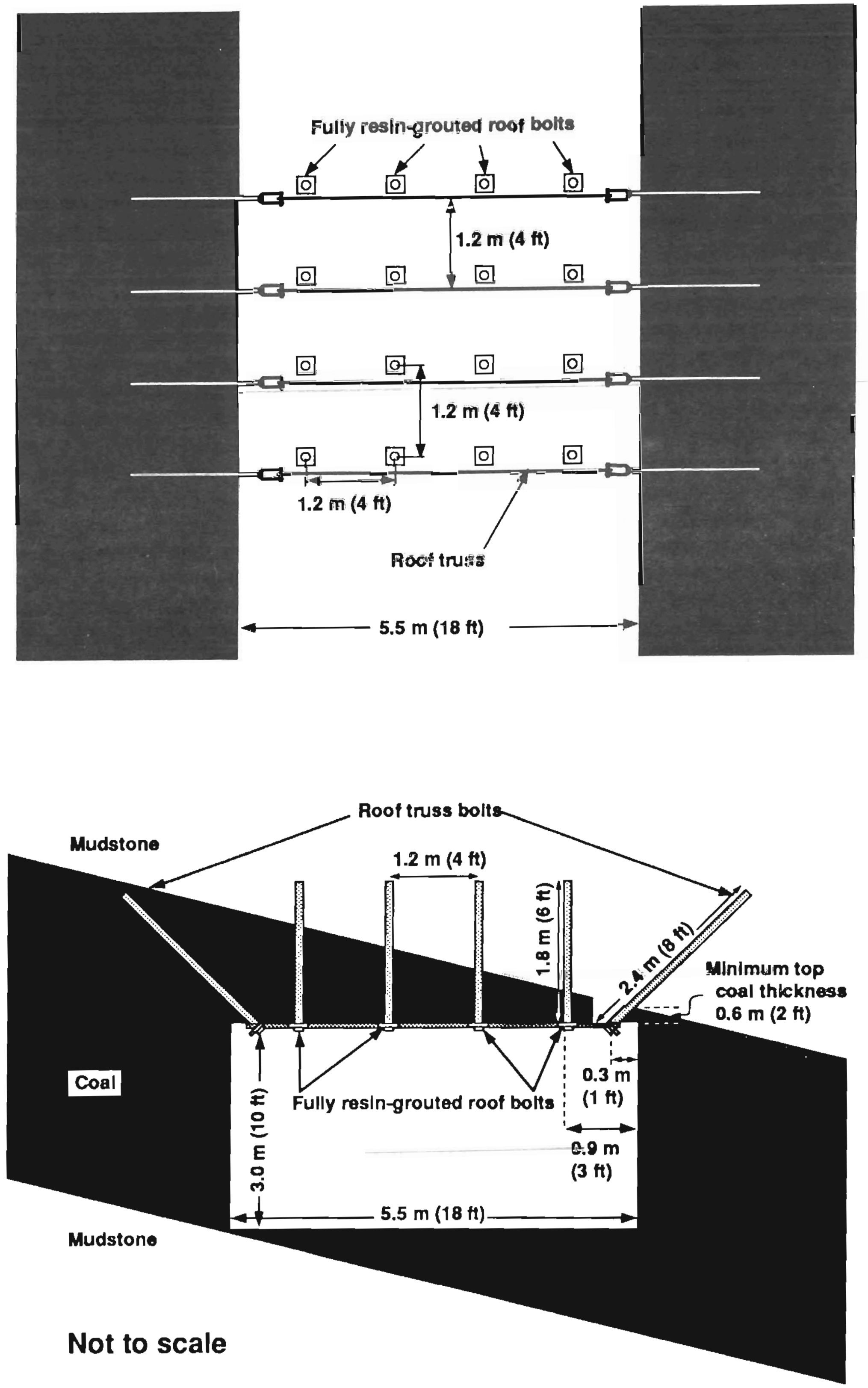
Longwall mining area of Shoshone No. 1 Mine during 1980-94, showing locations of drill holes DH94 and DH104 and USBM study site. (Courtesy Cyprus Shoshone Coal Corp.)

Figure 6



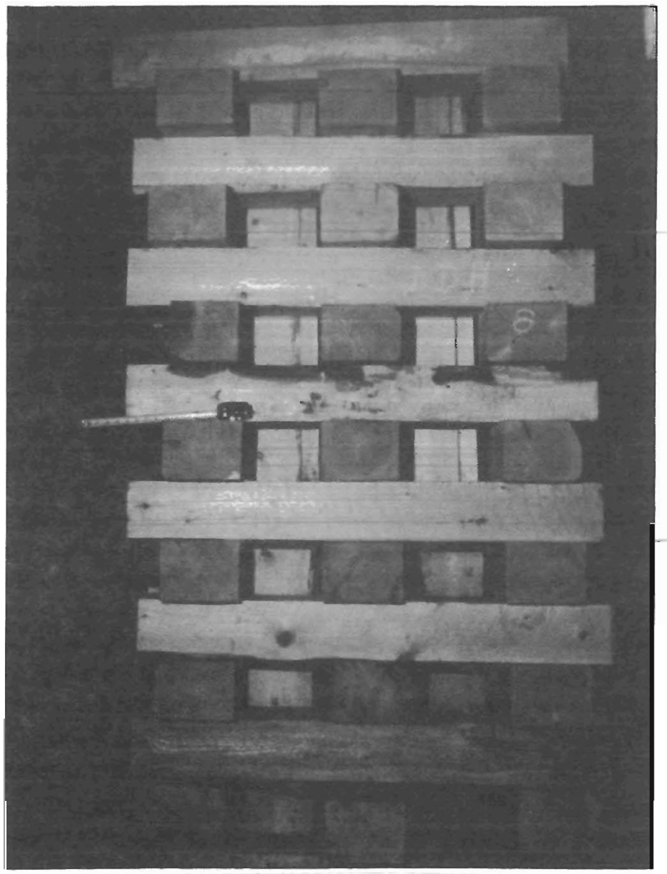
Overburden strata exposed in old strip mine highwall at Shoshone No. 1 Mine portals. Immediate roof mudstone lies between level of portals and bench across face of highwall. Thin siltstone layer forms road base of bench. Sandstone strata at about three-fourths of highwall height is lowest competent strata in main mine roof. (Photo: Liane Kadnuck, Denver Research Center, U.S. Bureau of Mines)

Figure 7



Primary roof support and entry configuration for Shoshone No. 1 Mine gate road entries. Top, plan view; bottom, cross section view.

Figure 8



Confined core wooden crib used for secondary support in tailgate entry of 11th Left longwall panel. (Photo: William Wuest, Denver Research Center, U.S. Bureau of Mines)

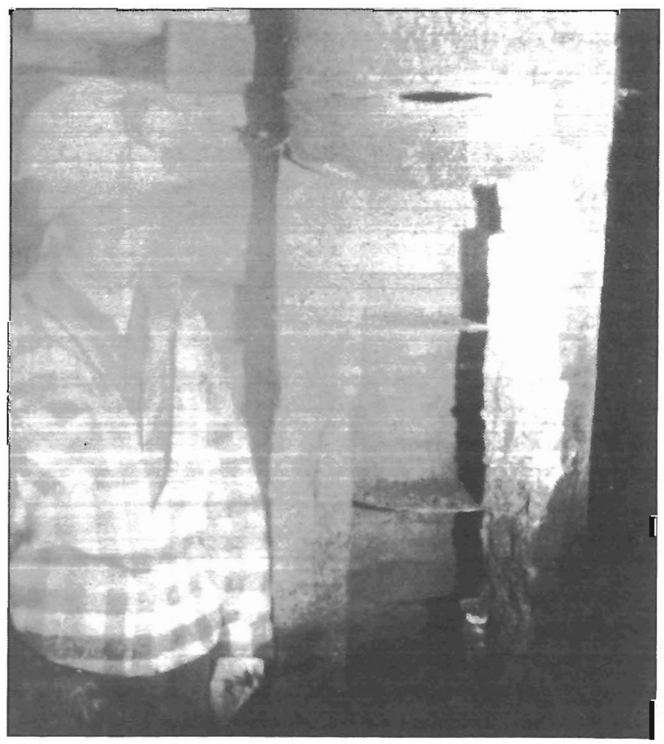
Figure 9

Steel beams with yieldable steel legs used for secondary gate-entry support. (Photo: Greg Molinda, Pittsburgh Research Center, U.S. Bureau of Mines)

Figure 10

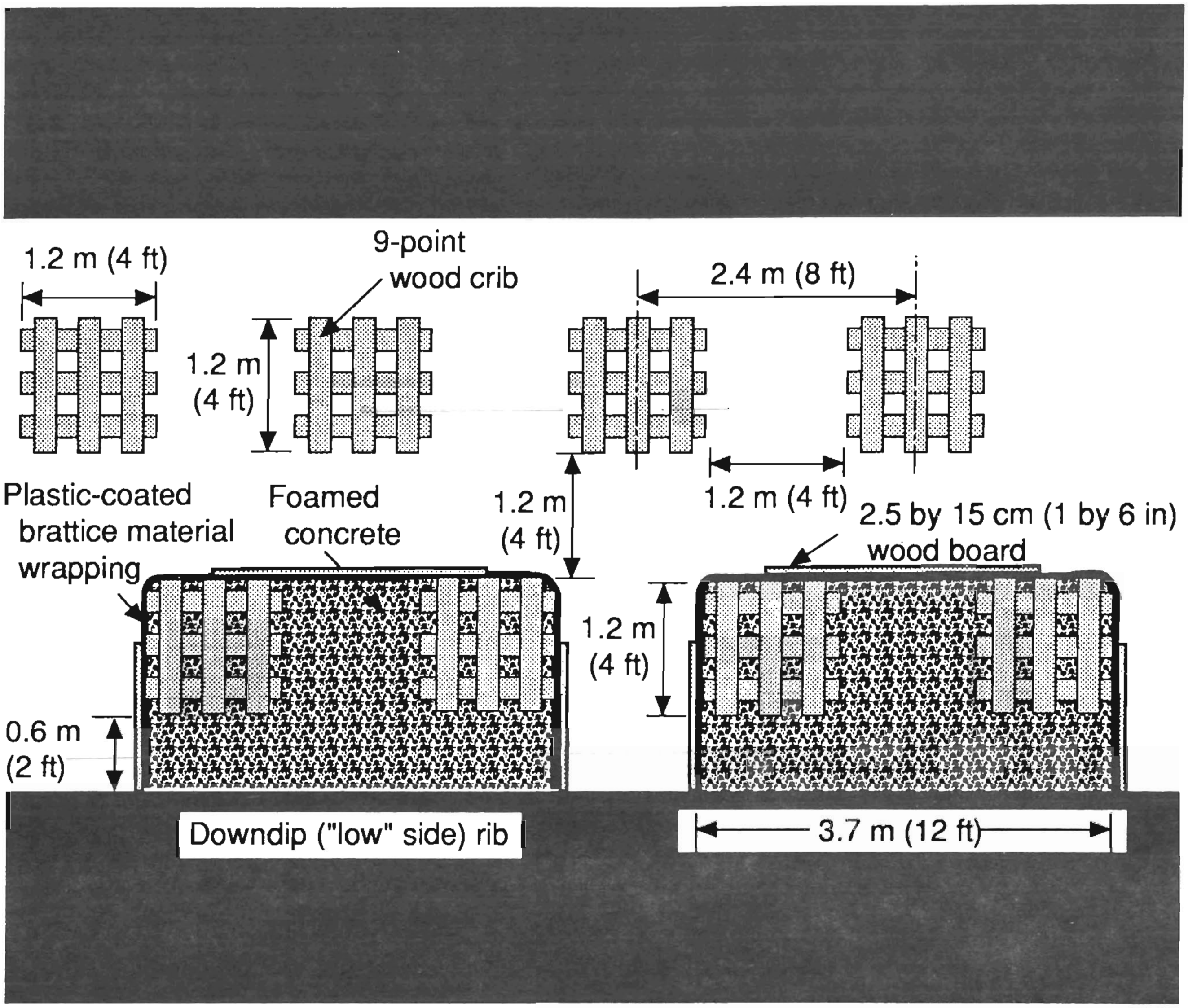
Concrete donut crib used for secondary support (on trial basis) in tailgate entry of 11th Left longwall panel. (Photo: William Wuest, Denver Research Center, U.S. Bureau of Mines)

Figure 11



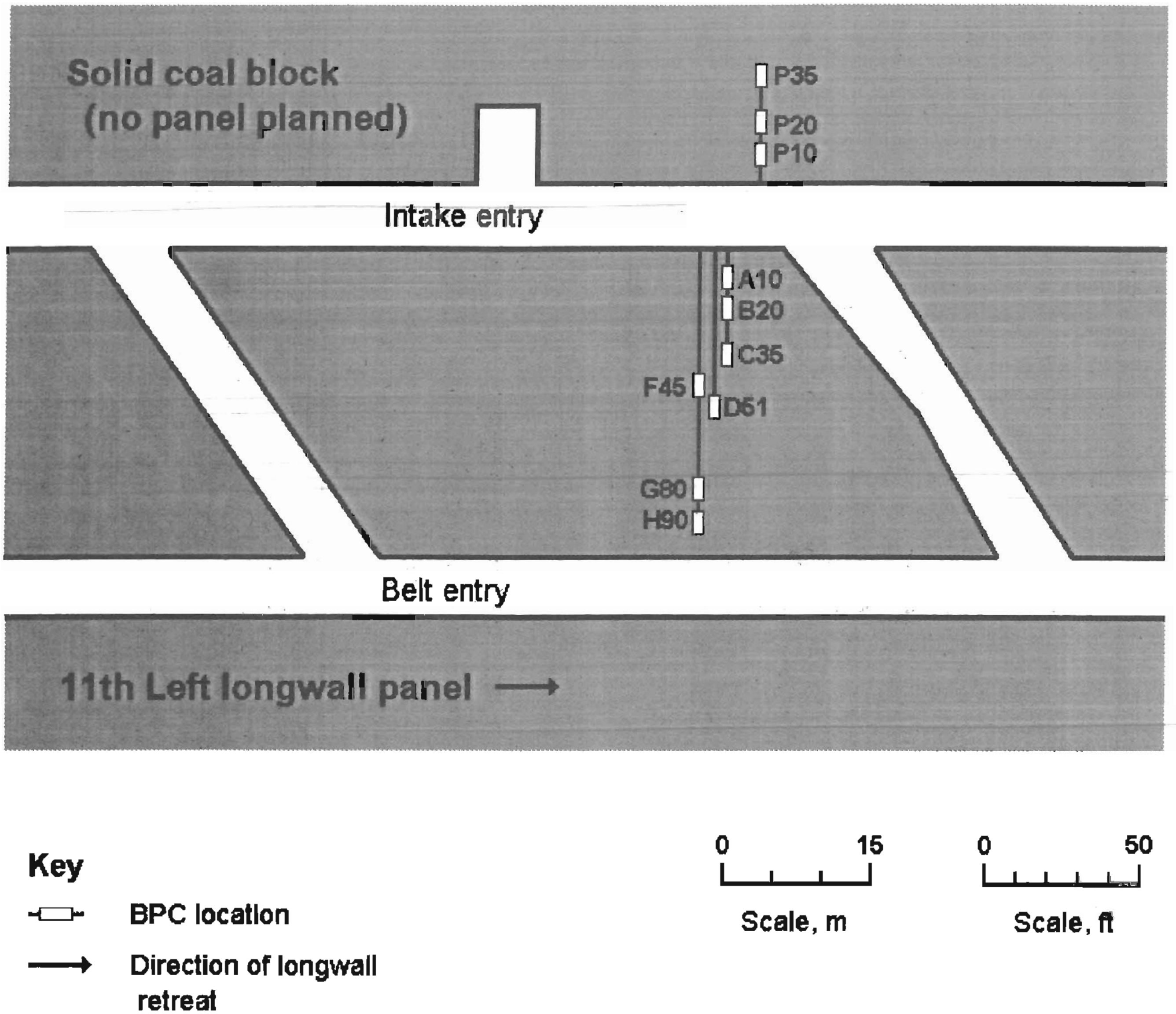
Concrete donut crib that has failed in compression and split vertically. (Photo: William Wuest, Denver Research Center, U.S. Bureau of Mines)

Figure 12



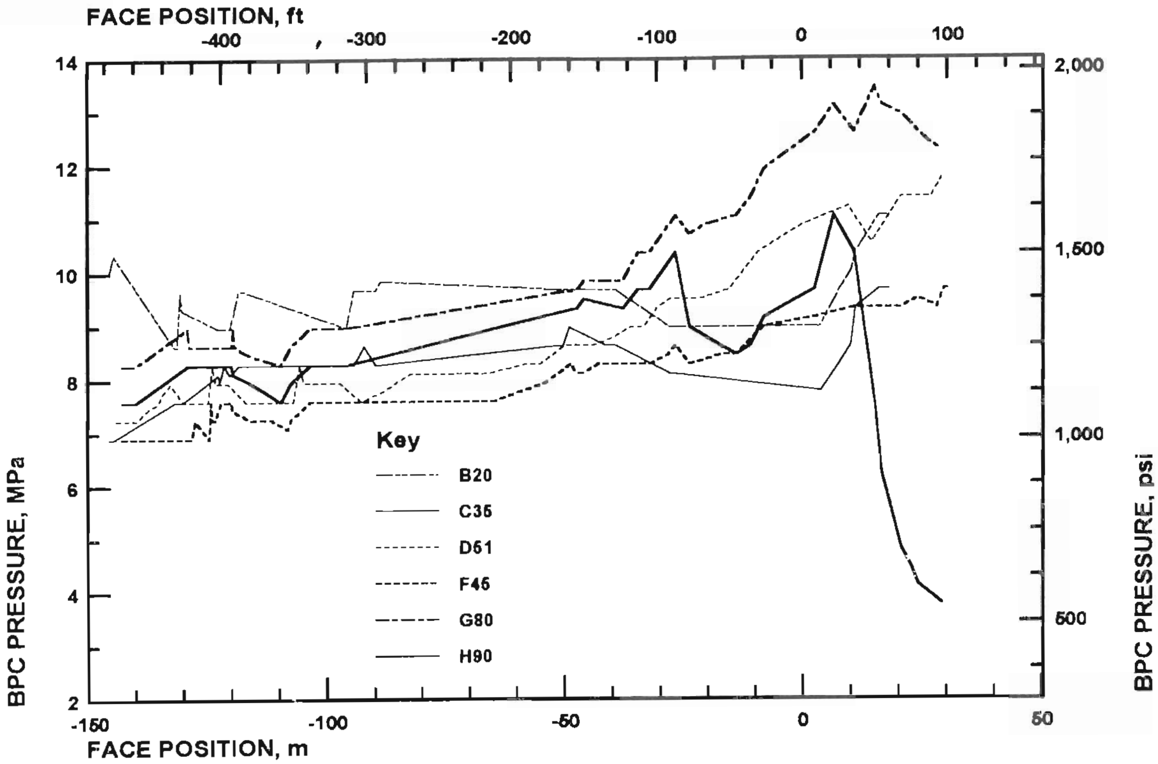
Cribbing pattern used for secondary support in tailgate entry of 11th Left longwall panel, showing configuration and position of wood-foamed concrete composite cribs.

Figure 13



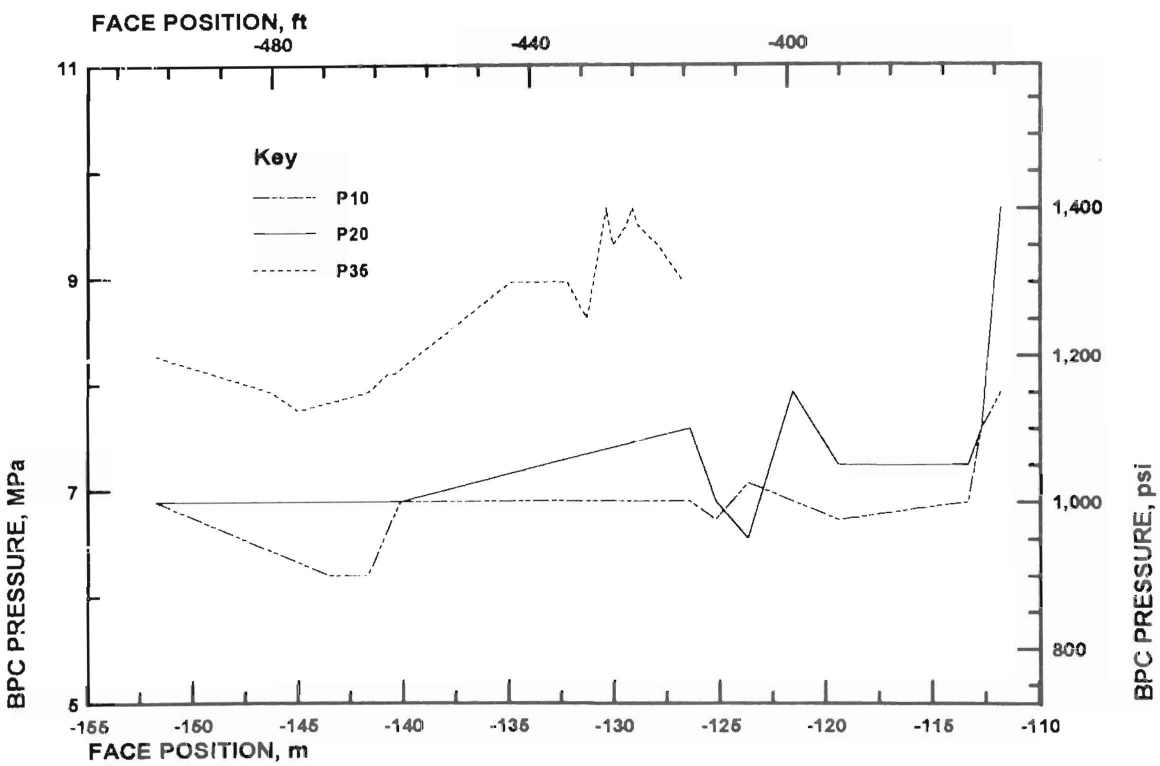
Study site instrumentation in 11th Left gate road.

Figure 14



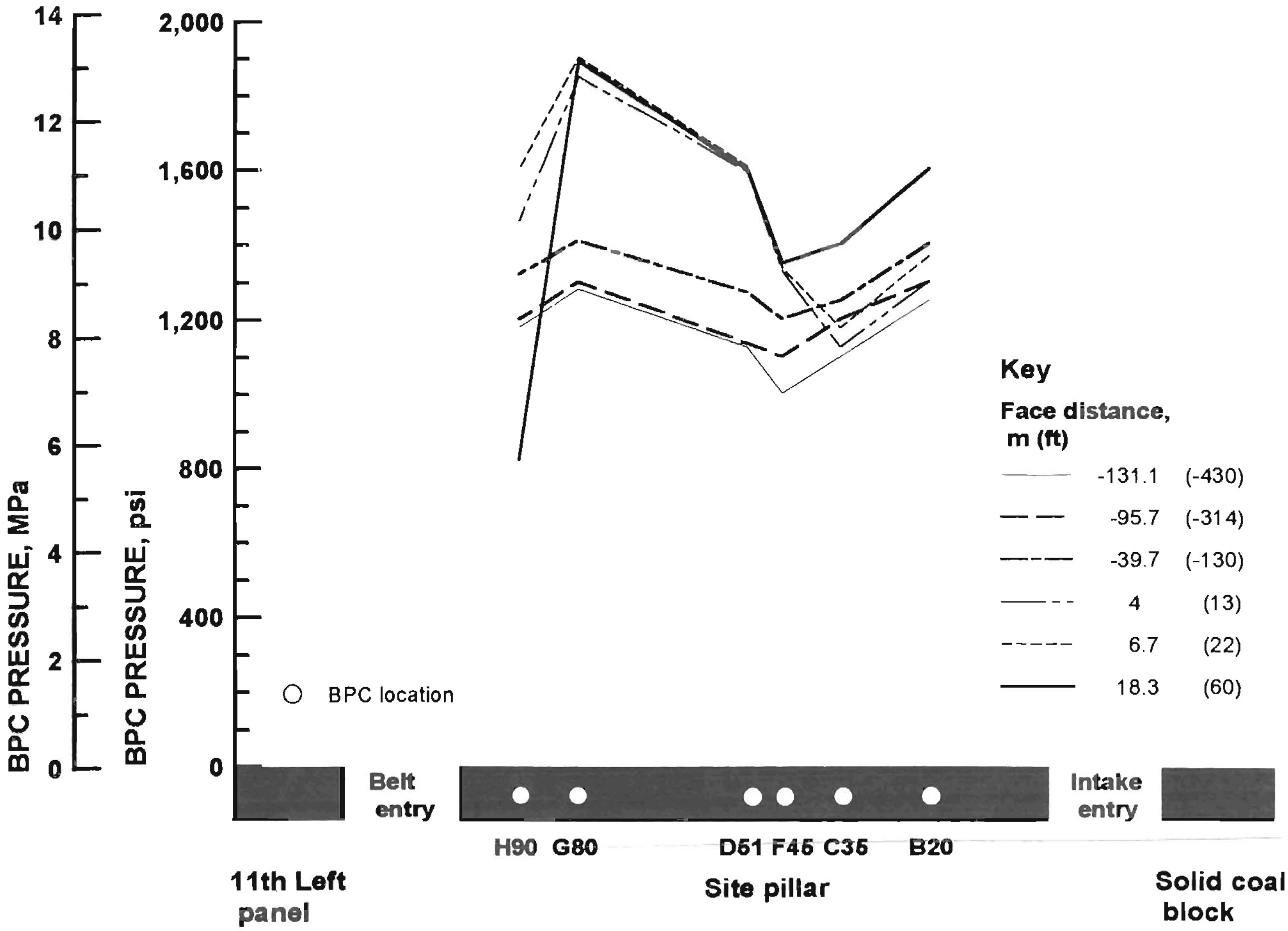
Vertical pressure relative to 11th Left panel face position at study site pillar.

Figure 15



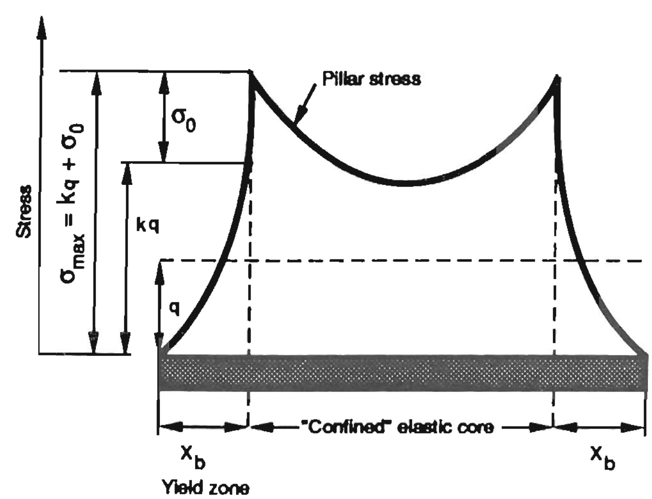
Vertical pressure relative to 11th Left panel face position at solid coal block opposite study site pillar.

Figure 16



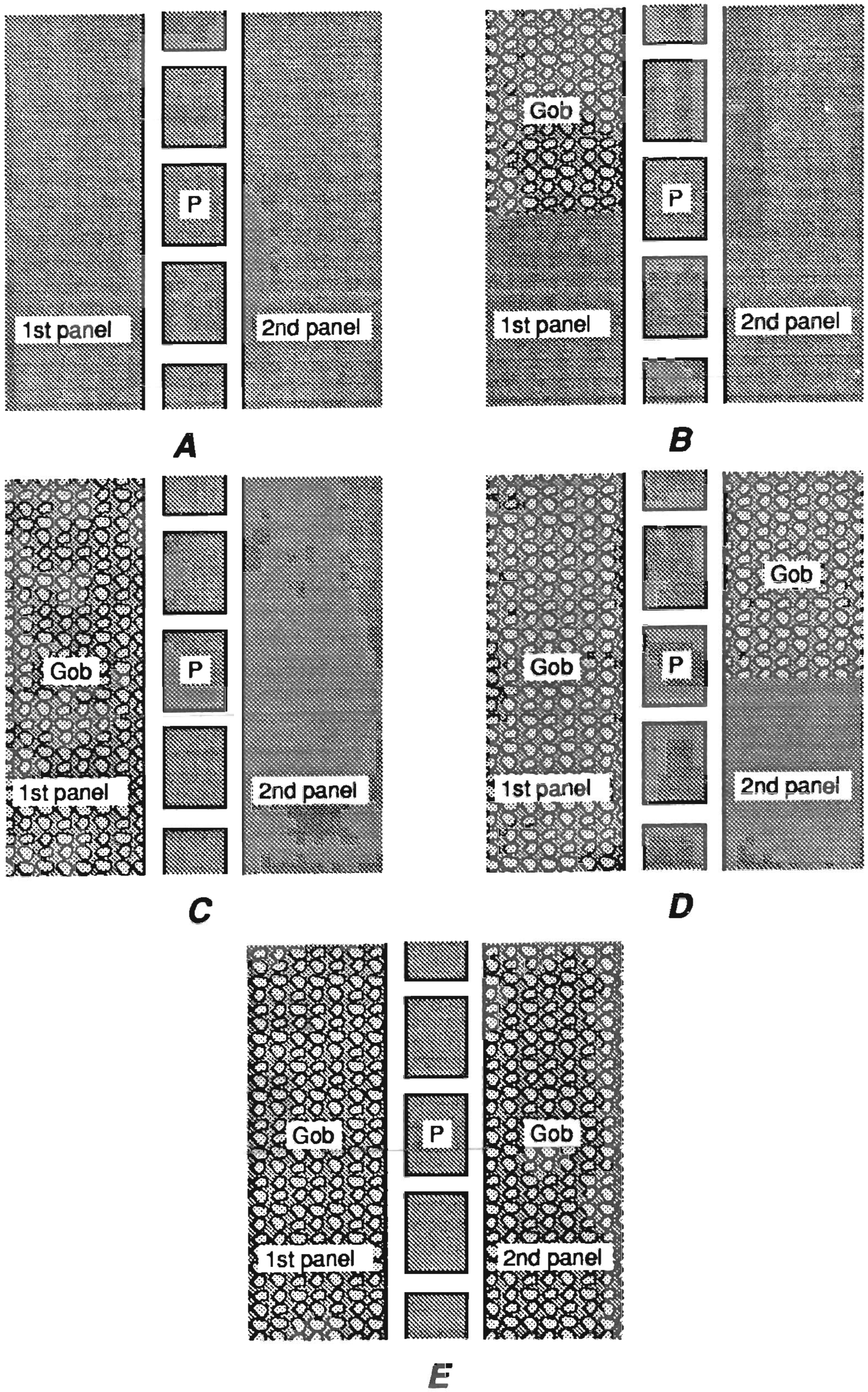
Vertical pressure profiles at site pillar for selected face positions (looking inby).

Figure 17



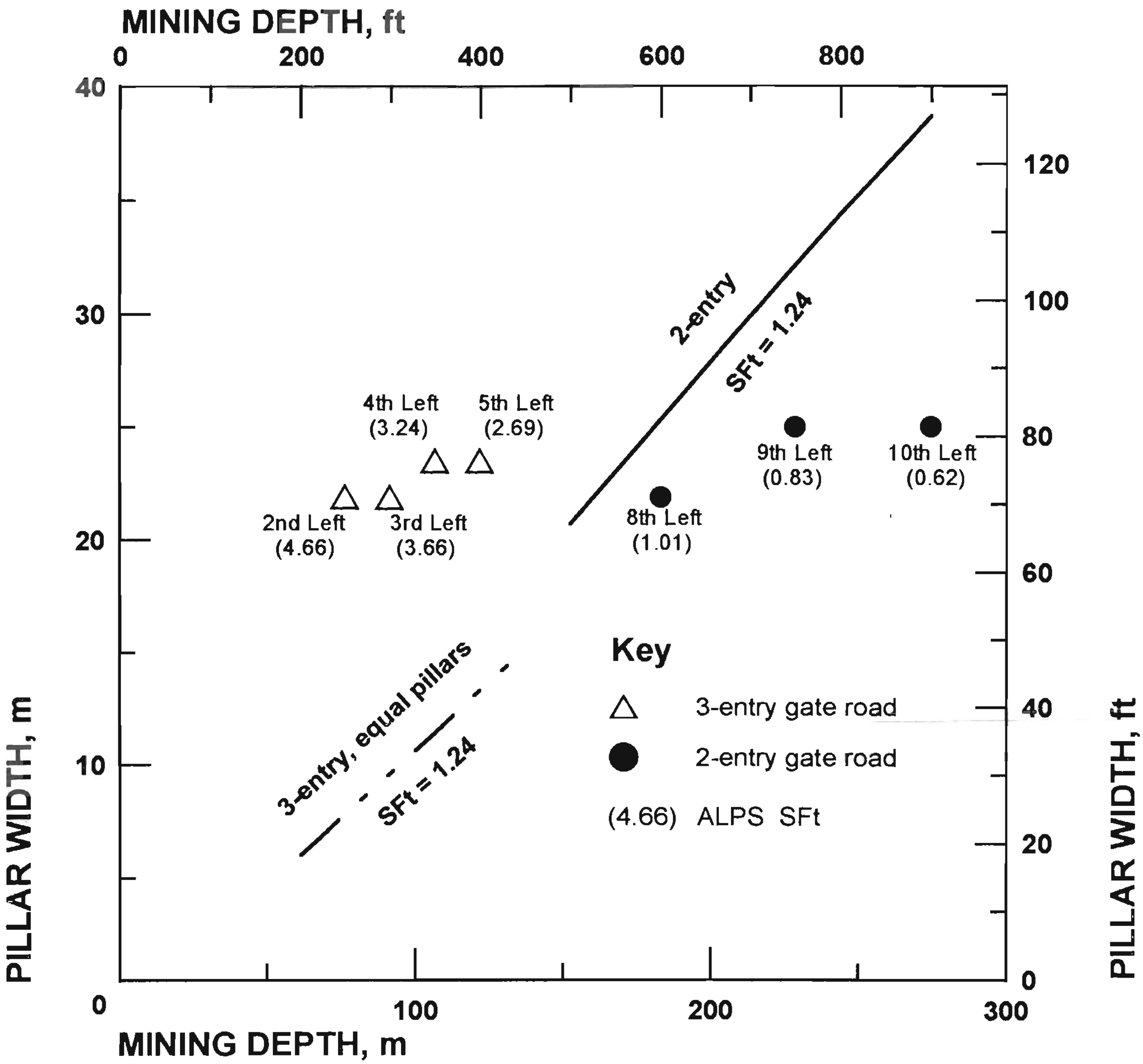
Conceptual vertical stress profile for Wilson's confined-core pillar. [From Carr and Wilson (25)]

Figure 18



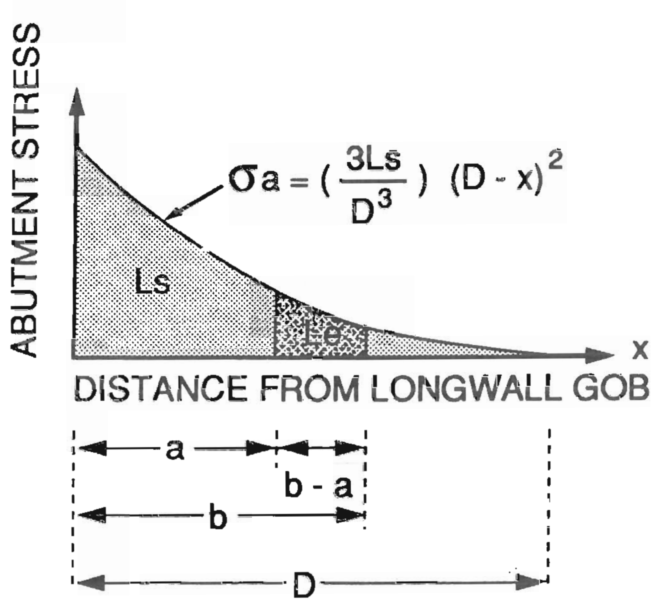
Sequence of ALPS loading conditions for pillar P in two-entry gate road. A, Development loading; B, headgate loading; C, bleeder loading; D, tailgate loading; E, isolated loading.

Figure 19



Chain pillar width and ALPS SFt of second-panel-mining tailgate roads for Shoshone No. 1 compared to equation 5 criterion for 0.61-m (2-ft) top coal thickness (CMRR = 37).

Figure 20

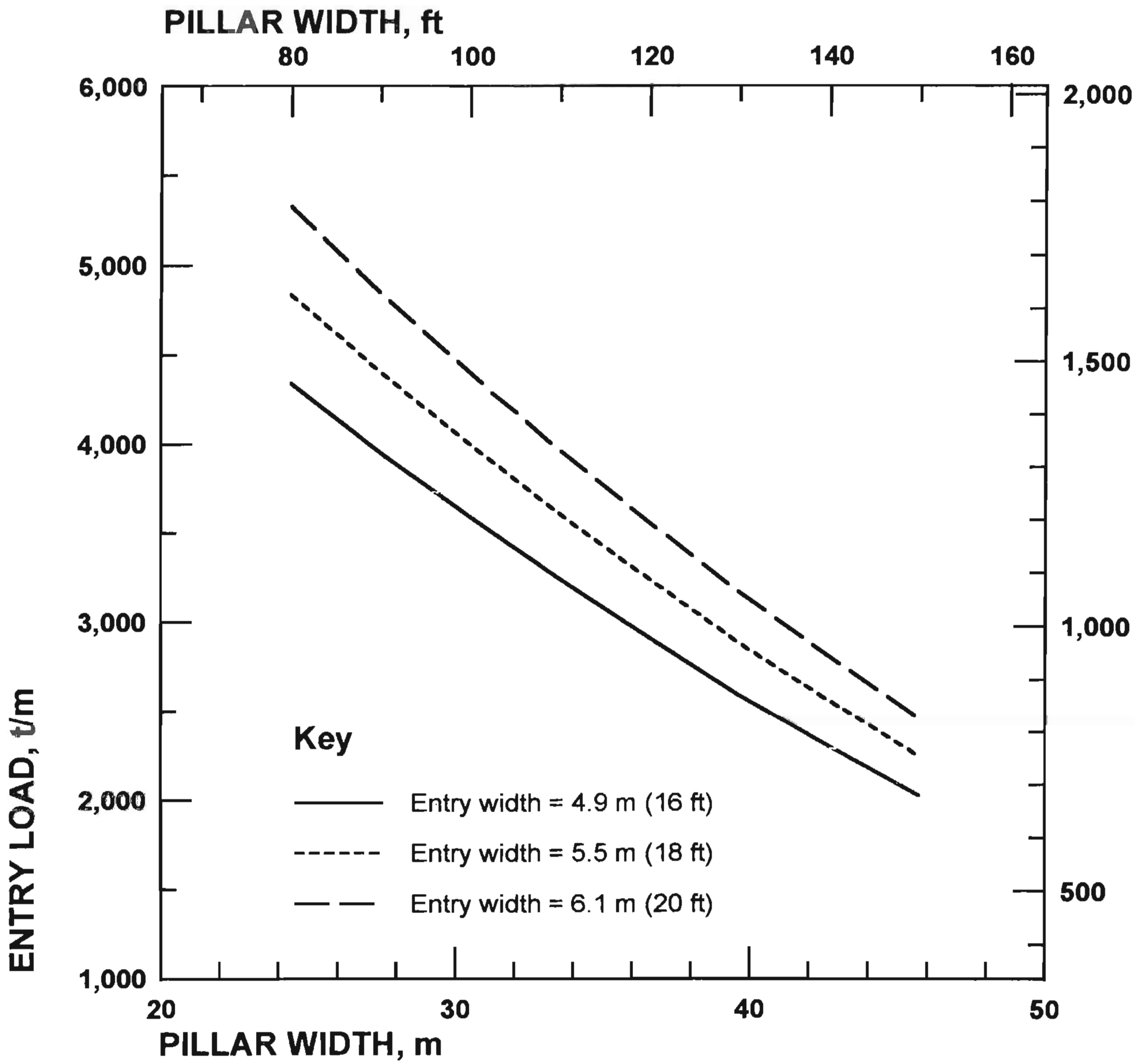


Key

σa	Abutment stress distribution function
Le	Abutment load on entry
D	Extent of side abutment influence zone
x	Distance from edge of longwall gob
Ls	Total side abutment load
a	Width of chain pillar in 2-entry gate road
b - a	Width of entry
b	Distance from edge of longwall gob to tailgate rib of next longwall panel

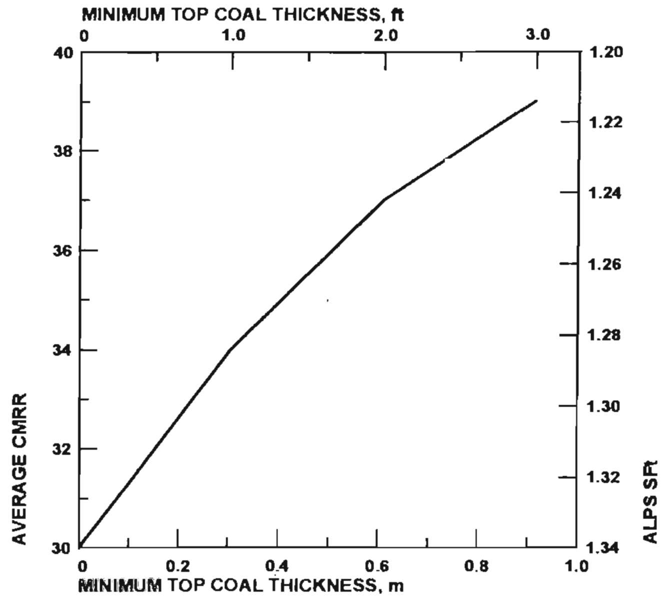
Distribution of side abutment load for two-entry gate road

Figure 21



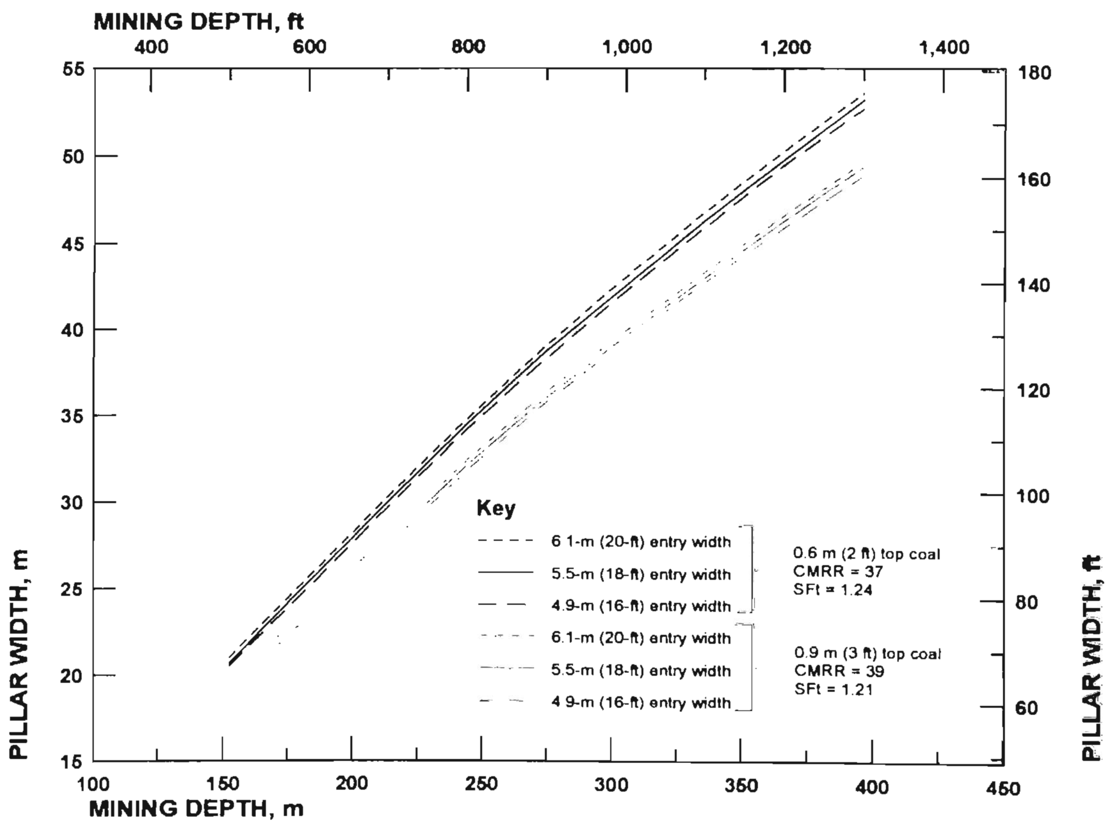
Hypothetical side abutment loading from first-panel mining (ALPS SFb) on second-panel tailgate entry in two-entry gate road. Mining depth is 335 m (1,100 ft).

Figure 22



Average CMRR and ALPS SFt (from equation 5) relative to minimum top coal thickness for Shoshone No. 1 Mine gate roads.

Figure 23



Results from ALPS-CMRR evaluation for effects of top coal thickness and entry width on two-entry gate road stability for Shoshone No. 1 Mine.

INT.BU.OF MINES,PGH.,PA 30178

☆ USGPO 709-009/20,067